TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XXV.

FEBRUARY, 1895, TO OCTOBER, 1895,
INCLUSIVE.

NEW YORK CITY:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1896.

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DISCUSSIONS.

OFFICERS.

For year ending February, 1896.*

President.

JOSEPH D. WEEKS, Pittsburgh, Pa.

Vice-Presidents.

J. F. HOLLOWAY,	New York City.
J. C. PLATT,	Waterford, N. Y.
E. V. D'INVILLIERS,	Philadelphia, Pa.

(Term expires February, 1896.)

WHEATON B. KUNHARDT,	New York City.
JAMES F. LEWIS,	Chicago, Ill.
CHARLES A. STETEFELDT,†	Oakland, Cal.

(Term expires February, 1897.)

Managers.

H. H. CAMPBELL,	Steelton, Pa.
W. L. SHEAFER,	Pottsville, Pa.
A. M. SHOOK,	Tracy City, Tenn.

(Term expires February, 1896.)

T. A. RICKARD,	Denver, Colo.
H. O. HOFMAN,	Boston, Mass.
JOHN A. CHURCH,	New York City.

(Term expires February, 1897.)

LEVI HOLBROOK,	New York City.
ALBERT R. LEDOUX,	New York City.
WILLIAM R. WEBSTER,	Philadelphia, Pa.

(Term expires February, 1898.)

Secretary.

R. W. RAYMOND,

13 Burling Slip, New York City

Treasurer.

THEODORE D. RAND,

Philadelphia, Pa.

Honorary Members.

PROF. RICHARD ÅKERMAN, Stockholm, Sweden; SIR LOWTHIAN BELL, Middlesborough, England; PROF. A. DAUBRÉE,‡ Paris, France; DR. THOMAS M. DROWN, South Bethlehem, Pa.; PROF. HATON DE LA GOUPILLIÈRE, Paris, France; PROF. HANNS HOEFER, Leoben, Austria; DR. BRUNO KERL, Berlin, Germany; PROF. JOSEPH LECONTE, Berkeley, Cal.; PROF. J. P. LESLEY, Philadelphia, Pa.; PROF. THEODOR RICHTER, Freiberg, Saxony; BERGRATH ALBERT SERLO, Berlin, Germany; PROF. PETER RITTER V. TUNNER, Leoben, Austria; PROF. DR. HERMANN WEDDING, Berlin, Germany.

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* The following officers were elected at the Annual Meeting, February, 1896: *President*, E. G. Spilsbury, Trenton, N. J.; *Vice-Presidents* (to serve two years), H. S. Chamberlain, Chattanooga, Tenn., Anton Eilers, Pueblo, Colo., Charles Kirchhoff, New York City; *Managers* (to serve three years), James Gayley, Pittsburgh, Pa., James F. Kemp, New York City, Benj. Smith Lyman, Philadelphia, Pa.; *Treasurer*, Theodore D. Rand, Philadelphia, Pa.; *Secretary*, Rossiter W. Raymond, New York City.

† Died March, 1896.

‡ Died May, 1896.

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO FEBRUARY, 1896.

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VI.	Pittsburgh, Pa.,	October, 1872,	i.	25
VII.	Boston, Mass.,	February, 1873,	i.	28
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IX.	Easton, Pa.,	October, 1873,	ii.	7
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XII.	Hazleton, Pa.,	October, 1874,	iii.	8
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XVI.	Washington, D. C.,	February, 1876,	iv.	18
XVII.	Philadelphia, Pa.,†	June, 1876,	v.	3
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XXV.	Baltimore, Md.,*	February, 1879,	vii.	217
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XXVII.	Montreal, Canada,	September, 1879,	viii.	121
XXVIII.	New York, N. Y.,*	February, 1880,	viii.	275
XXIX.	Lake Superior, Mich.,	August, 1880,	ix.	1
XXX.	Philadelphia, Pa.,*	February, 1881,	ix.	275
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XXXII.	Harrisburg, Pa.,	October, 1881,	x.	119
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XXXIV.	Denver, Col.,	August, 1882,	xi.	1
XXXV.	Boston, Mass.,*	February, 1883,	xi.	217
XXXVI.	Roanoke, Va.,	June, 1883,	xii.	3
XXXVII.	Troy, N. Y.,	October, 1883,	xii.	175
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XLI.	New York, N. Y.,*	February, 1885,	xiii.	585

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

Number.	Place.	Date.	Transactions.	
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* Annual meeting for the election of officers.

† Begun in February at New York City, for the election of officers, and adjourned to Florida.

PUBLICATIONS.

The publications of the Institute comprise:

Pamphlets.

1. The minutes of the Proceedings of each Meeting.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. (In nearly all cases in which papers, the titles of which appear in the Proceedings, are not subsequently published, they have been withdrawn by the authors.) These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the edition distributed without charge to members and associates not in arrears, a small supply is retained to meet subsequent demand. There are no copies on hand of papers read before 1880. The stock is nearly complete from 1880. These papers are for sale at the office of the Secretary, or are sent to purchasers by mail or express, charges paid, on receipt of the price, as follows:

No. of Pages.	Single Copies.	10 Copies.	20 Copies.
8 or less	\$0 06	\$0 30	\$0 50
8 to 12 inclusive	0 08	0 60	1 00
12 to 16 "	0 12	1 00	1 50
16 to 20 "	0 16	1 25	2 00
20 to 24 "	0 20	1 50	2 50
Over 24 subject to special price	......	......	......

Transactions.

The volumes of *Transactions*, which are published annually, contain the list of officers, rules, etc., the Proceedings, and the papers revised for final publication. (In this revision, after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind, by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there is no essential change, the correction of typographical errors and additions of later information being the usual alterations.)

The twenty-four volumes of *Transactions* now published, are for sale as follows:

Vols. I. to IV., inclusive, each, ... $2 00
Vols. V. to VIII., inclusive, each, ... 3 00
Vol. IX., ... 5 00
Vol. X. (a small supply only on hand), ... 10 00
Vols. XI. to XXV., inclusive, each, ... 5 00

These prices are for paper covers. Half-morocco binding $1 extra per volume.

INDEXES AND SPECIAL EDITIONS.

Index, Vols. I. to XV., inclusive, of the *Transactions*, cloth, . $1 00
half-morocco, ... 1 50
Index, Vols. XVI. to XX., inclusive, paper, ... 1 00
Indexes, Vols. I. to XV. and XVI. to XX., bound in one volume, cloth, ... 2 00
half-morocco, ... 2 50
Technical Education, being papers and discussions of the Philadelphia Meeting, 1876 (not all printed in the *Transactions*), paper, ... 50
Steel Rails, papers by Messrs. Sandberg, Dudley, and Holley, with discussions at meetings in 1881, paper, . . 50
Geological Map of the United States, colored after the scale proposed by the International Geological Congress, by Prof. C. H. Hitchcock, ... 1 00
Memorial of Alexander L. Holley, with portrait, cloth, . 1 00
The Genesis of Ore-Deposits, by Prof. Franz Posepny (with discussions and portrait), cloth, ... 2 50
List of Members, Rules, etc., paper, ... 25

AUTHORS' EDITIONS OF PAMPHLETS.

Extra copies, when ordered before the types have been distributed, are furnished to authors, under Rule VII., at the following rates:

No. of Pages.	50 Copies.	100 Copies.	250 Copies.	Each additional 100 copies above 250.
4 or less	$1 25	$1 50	$2 25	$0 50
4 to 8 inclusive	1 75	2 25	3 25	0 75
8 to 12 "	2 25	3 00	4 25	1 00
12 to 16 "	2 75	3 75	5 25	1 25
16 to 20 "	3 25	4 50	6 25	1 50
20 to 24 "	3 75	5 25	7 25	1 75
24 to 28 "	4 25	6 00	8 25	2 00
28 to 32 "	4 75	6 75	9 25	2 25
Covers (including printing on first page of the same), extra	1 50	2 00	3 00	0 50

When a paper contains one or more *separate* plates or "folders," these will be charged in reprinting as follows: One page or one fold, the same as four pages in the above table; each additional fold, the same as four additional pages. These prices are for plates on the ordinary paper, used in the edition issued "subject to revision." If special bank-note paper is desired, such as is used in the Volumes of *Transactions*, the price for the plates will be doubled.

All communications and remittances should be addressed to R. W. Raymond, Secretary, P. O. Box 223, New York City.

RULES

ADOPTED MAY, 1873. AMENDED MAY, 1875, 1877, AND 1878, FEBRUARY, 1880, 1881, 1887, AND 1890.

I.

OBJECTS.

THE objects of the AMERICAN INSTITUTE OF MINING ENGINEERS are to promote the arts and sciences connected with the economical production of the useful minerals and metals, and the welfare of those employed in these industries, by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate, by means of publications among its members and associates, the information thus obtained.

II.

MEMBERSHIP.

The Institute shall consist of Members, Honorary Members, and Associates. Members and Honorary Members shall be professional mining engineers, geologists, metallurgists, or chemists, or persons practically engaged in mining, metallurgy, or metallurgical engineering. Associates shall include all suitable persons desirous of being connected with the Institute, and duly elected as hereinafter provided. Each person desirous of becoming a member or associate shall be proposed by at least three members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) upon receiving three-fourths of the votes cast, and shall become a member or associate on the payment of his first dues. Each person proposed as an honorary member shall be recommended by at least ten members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) on receiving nine-tenths of the votes cast; *Provided*, that the number of honorary members shall not exceed twenty. The Council may at any time change the classification of a person elected as associate, so as to make him a member, or *vice versa*, subject to the approval of the Institute. All members and associates shall be equally entitled to the privileges of membership; *Provided*, that honorary members shall not be entitled to vote.

Any member or associate may be stricken from the list on recommendation of the Council, by the vote of three-fourths of the members and associates present at any annual meeting, due notice having been mailed in writing by the Secretary to the said member or associate.

III.

DUES.

The dues of members and associates shall be ten dollars, payable upon their election, and ten dollars per annum thereafter, payable in advance at the annual meeting. Honorary members shall not be liable to dues. Any member or associate not in arrears may become by the payment of one hundred dollars at one time a life-member or associate, and shall not be liable thereafter to annual dues. Any member or associate in arrears may, at the discretion of the Council, be deprived of the receipt of publications, or stricken from the list of members when in arrears for one year; *Provided*, that he may be restored to membership by the Council on payment of all arrears, or by re-election after an interval of three years.

IV.

OFFICERS.

The affairs of the Institute shall be managed by a Council, consisting of a President, six Vice-Presidents, nine Managers, a Secretary and a Treasurer, who shall be elected from among the members and associates of the Institute at the annual meetings, to hold office as follows:

The President, the Secretary, and the Treasurer for one year (and no person shall be eligible for immediate re-election as President who shall have held that office subsequent to the adoption of these rules, for two consecutive years), the Vice-Presidents for two years, and the Managers for three years; and no Vice-President or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. At each annual meeting a President, three Vice-Presidents, three Managers, a Secretary and a Treasurer shall be elected, and the term of office shall continue until the adjournment of the meeting at which their successors are elected.

The duties of all officers shall be such as usually pertain to their offices, or may be delegated to them by the Council or the Institute; and the Council may in its discretion require bonds to be given by the Treasurer. At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Vacancies in the Council may occur by death or resignation; or the Council may, by a vote of the majority of all its members, declare the place of any officer vacant, on his failure for one year, from inability or otherwise, to attend the Council meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *Provided*, that the said appointment shall not render him ineligible at the next annual meeting.

Five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or business may be transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to the approval of a majority of the Council, subsequently given in writing to the Secretary, and recorded by him with the minutes.

V.

ELECTIONS.

The annual election shall be conducted as follows: Nominations may be sent in writing to the Secretary, accompanied with the names of the proposers, at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before the said meeting, mail to every member or associate (except honorary members), a list of all the nominations for each office so received, together with a copy of this rule, and the names of the persons ineligible for election to each office; and if the Council, or a Committee thereof, appointed for the purpose, shall have recommended any nominations, such recommendation may also be sent to members and associates with the said list of all nominations made, but not upon the same paper. And each member or associate, qualified to vote, may vote, either by striking from or adding to the names of the said list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing said altered or prepared ballot with his name, and either mailing it to the Secretary or presenting it in person at the annual meeting; *Provided*, that no member or associate in arrears since the last annual meeting shall be allowed to vote until the said arrears shall have been paid. The ballots shall be received and examined by three Scrutineers, appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected, and the Scrutineers shall so report to the presiding officer. The ballots shall be destroyed, and a list of the elected officers, certified by the Scrutineers, shall be preserved by the Secretary.

VI.

MEETINGS.

The annual meeting of the Institute shall take place on the third Tuesday of February, at which a report of the proceedings of the Institute and an abstract of the accounts shall be furnished by the Council. Two other regular meetings of the Institute shall be held in each year, at such times and places as the Council shall select, and notice of all meetings shall be given by mail, or otherwise, to all members and associates, at least twenty days in advance. Special meetings may be called whenever the Council sees fit; and the Secretary shall call a special meeting on a requisition signed by fifteen or more members. The notices for special meetings shall state the business to be transacted, and no other shall be entertained.

Every question which shall come before any meeting of the Institute, shall be decided, unless otherwise provided by these Rules, by the votes of a majority of the members then present. Any member or associate may introduce a stranger to any meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

VII.

PAPERS.

The Council shall have power to decide on the propriety of communicating to the Institute any papers which may be received, and they shall be at liberty, when they think it desirable, to direct that any paper read before the Institute, shall be printed in the Transactions. Intimation, when practicable, shall be given, at each general meeting, of the subject of the paper or papers to be read, and of the questions for discussion at the next meeting. The reading of papers shall not be delayed beyond such hour as the presiding officer shall think proper; and the election of members or other business may be adjourned by the presiding officer, to permit the reading and discussion of papers.

The copyright of all papers communicated to, and accepted by, the Institute, shall be vested in it, unless otherwise agreed between the Council and the author. The author of each paper read before the Institute shall be entitled to twelve copies, if printed, for his own use, and shall have the right to order any number of copies at the cost of paper and printing, provided said copies are not intended for sale. The Institute is not, as a body, responsible for the statements of fact or opinion advanced in papers or discussions at its meetings, and it is understood that papers and discussions should not include matters relating to politics or purely to trade.

VIII.

AMENDMENTS.

These Rules may be amended at any annual meeting by a two-thirds vote of the members present; *Provided*, that written notice of the proposed amendment shall have been given at a previous meeting; *and Provided, also*, that the amendment or amendments so adopted shall be printed upon a ballot and sent, not later than the next distribution of printed matter, to all members and associates not in arrears for the preceding year (except honorary members and foreign members elected before February, 1880), and each person receiving the same shall be requested to return it to the Secretary with his written vote of Yes or No to each amendment, and his signature; and the President shall appoint as scrutineers three members or associates, who shall examine all of the said ballots which shall have been returned within one month from the date of their distribution, and shall report the result; and the Secretary shall publish and distribute to members, not later than the next distribution of printed matter, an announcement of the said result so reported, together with the text of the additional or amended rule or rules so adopted; and the amendment or amendments approved by the majority of the ballots so returned and reported shall become part of these rules from and after the publication of said announcement by the Secretary.

NOTE.—At the Annual Meeting of February, 1896, the following amendments to the foregoing rules were adopted for submission to the vote of members by mail, as provided in Rule VIII., and were, subsequently, finally adopted by such vote and duly announced as part of the rules. The rules will be printed as thus amended in vol. xxvi. (containing the Proceedings of the said Annual Meeting).

AMENDMENTS.

1. Rule II. After the words "shall not be entitled to vote," add: *and members or associates whose Post-Office address shall be outside of the United States, Canada and Mexico shall not be entitled to vote by mail, except upon proposed amendments to the Rules.*

2. Rule III. Strike out: *at the annual meeting;* and substitute: *on the first day of each calendar year.*

3. Rule VI. Strike out provision for special meetings and make second sentence read: *Other meetings shall he held at such times and places as the Council shall select, and notice of all meetings shall be given,* etc.

4. Rule VII. Make title *Papers and Publications.* Insert after first paragraph: *The published papers and volumes of Transactions shall be distributed to all members and associates not in arrears, and may be sold to the public upon such conditions as the Council shall prescribe; but the Council may, in its discretion, omit sending to members and associates outside of the United States, Canada and Mexico, special circulars unless the same contain proposed amendments to the Rules.*

5. Rule VII. Add: *Nor shall the Council or the Institute officially approve or disapprove any technical or scientific opinion or any proposed enterprise outside the management of the meetings, discussions and publications of the Institute as provided in these Rules:* Provided, however, *that committees may be appointed by the Council or the Institute to make investigations and submit reports at meetings of the Institute; but no action shall be taken binding the Institute for or against the conclusions of any such reports.*

Proceedings of the Sixty-Eighth (Twenty-Fifth Annual) Meeting, New York and Florida, February and March, 1895.

New York Session.

The opening session was held at the office of the Secretary on Tuesday, February 19th, at 12 o'clock, President Fritz in the chair.

Messrs. E. G. Spilsbury, Charles Kirchhoff and Theodore Dwight were appointed as Scrutineers, to examine the ballots for officers received, and to report the result at a subsequent session.

The Annual Report of the Council was presented, as follows:

Annual Report of the Council.

In accordance with the rules, the Council makes the following report to the Institute:

The financial statement of the Secretary and Treasurer shows receipts from all sources for the year ending February 1st (including $1699.96, on hand at the beginning of the year), of $29,001.99, and expenditures of $27,800.22, leaving a surplus of $1201.77, being a reduction in the surplus of February 1, 1894, of $498.19. In addition to this, the Treasurer holds U. S. bonds of the par value of $2900 and a special deposit of $4598, proceeds of U. S. bonds called in and paid by the government, which fund has not been permanently re-invested. The detailed statement of receipts and expenditures is as follows:

Receipts.

Balance from statement, February 1, 1894,		$1,699.96
Annual dues,	$20,413.28	
Life membership,	1,000.00	
Binding of *Transactions*,	3,054.06	
Sale of volumes *Transactions*,	1,310.95	
" " pamphlets,	346.36	
Authors' pamphlets,	185.68	
Electrotypes,	195.17	
Interest on U. S. bonds and deposits,	392.13	
World's Fair fund,	400.00	
Miscellaneous,	4.40	27,302.03
		$29,001.99

Disbursements.

Printing Volumes XXII. and XXIII. *Transactions*, . .	$4,570.48	
" pamphlet edition of papers, ...	3,741.70	
" authors' edition, ...	92.75	
" mailing list, ...	116.00	
" circulars and ballots, ...	111.45	
Binding Volumes XXII. and XXIII. *Transactions*, . .	2,776.58	
" exchanges, ...	297.16	
Engraving and electrotyping, ...	1,012.00	
Postage, including P. O. box-rent, ...	1,184.37	
Stationery, ...	325.67	
Rent, including Treasurer's safe-deposit box, . . .	810.00	
Express and freight charges, ...	1,592.12	
Telephone, ...	219.53	
Telegrams, cablegrams and car fare, ...	15.96	
Coal, ice and porters, ...	174.37	
Salaries, including clerks and stenographers, . . .	9,395.48	
Storage of *Transactions*, ...	186.32	
Special stenographers and expenses of meetings, . . .	540.34	
Gas, ...	20.73	
Office supplies and repairs, ...	337.22	
Expenses mailing batches, etc., ...	110.55	
Insurance, ...	59.40	
Refunding over payments, ...	54.00	
Library additions, ...	15.74	
World's Fair, special expenses, ...	40.30	27,800.22
Balance, ...		1,201.77
		$29,001.99

This financial statement is deemed a subject for congratulation, in view of the peculiar circumstances. On the one hand, the expenses of the year included the cost of two volumes instead of one, besides increase in other items, resulting from the International meetings at Chicago in 1893, and thus amounted to $27,800.22—an amount which has never been equalled in the history of the Institute, except in 1890, when the expenses connected with the reception of the Iron and Steel Institute swelled the total to $32,057.56. On the other hand, the commercial depression which has prevailed during the year, and which has affected mining probably more than any other branch of industry, has made it exceptionally difficult for our members to pay their annual dues. The severity of these unfavorable conditions is indicated by the extraordinarily large number of the names on the Secretary's list of members seeking professional engagements, which includes many experts of long experience and high standing. And the same inference

may be drawn from the significant fact that there have been, during the year, about 1300 changes of address in the Institute list of less than 2500 members and associates. By far the greater part of these changes (many hundred in excess of the usual average) were due to the loss of positions by reason of "hard times."

By vigilant economy in expenditures and patient perseverance in the treatment of members in arrears, the necessary outlays of the year have been met, as the preceding statement shows, without borrowing from the reserve-fund.

The amount reported as received from sale of volumes of *Transactions* includes the amount of $263.34 received from the Columbian Committee of the Institute, and constituting the balance of the special fund in its hands. This balance was paid for presentation copies of Volumes XXII. and XXIII., containing the papers and proceedings of the International meetings at Chicago, which were distributed to the representative engineers and engineering societies participating in the International Engineering Congress.

The amount of $400, reported as received from the World's Fair Fund, was the unexpended balance of the contribution of $4000 from the Special Columbian Fund of the Institute to the maintenance of the Engineering Headquarters in Chicago, during the Exposition.

Two meetings were held during the year: one at Virginia Beach, Va., in February, and the other at Bridgeport, Conn., in October. The social and professional interest of both will remain impressed upon the memory of the large number who attended them; and Volume XXIV. of the *Transactions*, comprising about 1100 pages, now in press, will amply establish the value of the contributions embodied in papers and discussions.

Changes in membership have taken place during the year as follows: 157 members and 21 associates have been elected; 7 associates have become members; the deaths of 1 honorary member, 23 members and 3 associates have been reported; 35 members and 8 associates have resigned, and 93 members and 8 associates have been dropped for continued default in the payment of dues. These changes are tabulated as follows, showing a net gain of 6 in total membership:

	H. M.	F. M.	M.	A.	Totals.
At date of last report,	15	38	2204	174	2431
Gains: By Election,	...	...	157	21	178
Change of status,	...	...	7	...	7
Losses: By Resignation,	...	...	35	8	43
Dropping,	...	...	93	8	101
Change of status,	...	...	...	7	7
Death,	1	...	24	3	28
Total gains,	...	...	164	21	185
Total losses,	1	...	152	26	179
Present membership,	14	38	2216	169	2437

The list of deaths comprises the names of M. F. Gaetzschmann, honorary member, and the following members and associates: Carl Amsler (1886), Charles H. Bosher (1891), Amos Bowman (1886), Spruille Braden (1887), J. H. Bramwell (1871), F. J. Carrel (1887), R. Neilson Clark (1872), Charles D. Cowland (1891), Wallace H. Dodge (1884), E. B. Ely (1879), John D. Evans (1880), Edward G. Gilbert (1883), Max J. Hartung (1890), E. C. O. Huhn (1886), A. L. Inman (1876), R. H. Lamborn (1876), E. B. Leisenring (1882), George G. Lobdell (1891), Henry B. Nason (1883), James Neilson (1879), Charles O. Parsons (1874), H. L. Reed (1892), H. O. Reinhardt (1883), Elliott Roosevelt (1893), T. L. Skinner (1888), J. H. Striedinger (1893), and J. Wentz Wilson (1887).

While this catalogue includes the names of Mr. Bramwell, one of the founders of the Institute, of Mr. Clark, one of its early members, and of Messrs. Ely, Inman, Lamborn, Neilson and Parsons, all of whom joined before 1880, and continued to the end their interest and support, it is noticeable that more than two-thirds of those removed by death had become connected with the Institute in comparatively recent years. This is largely due, no doubt, to the circumstance that the large increase of membership since 1880 (when the total of all classes was but 792) offers a wider field for the exhibition of the effects of mortality. Moreover, the date of election to membership is not necessarily an indication of the age of a member. Yet in this respect it may be fairly said that our early members were, on the average, more advanced in years than the recruits since received from among the graduates of technical schools and the practitioners of a second generation. At all events, while we lament the loss of more recent associates, we may naturally be

grateful for the numerous comrades of many years still spared to us.

The meeting was then adjourned, to be resumed at Ocala, Florida, on Wednesday, March 27th.

FLORIDA SESSIONS.

LOCAL COMMITTEES.

Ocala.—Mayor John G. Reardon, Edward Holder, W. S. Proskey, G. M. Wells, George Mackay, C. S. Clarke, W. W. Pickford, E. C. Bird, L. R. Chazal.

Silver Springs.—(Ladies' Committee) Mesdames John G. Reardon, E. W. Agnew, R. B. McConnell, Richard McConathy, John H. Burchell, Frank E. Harris, W. K. McDonald, Miss Annie Martin, Miss Riché, Mr. W. S. Proskey, *Chairman.*

Tampa.—J. H. King, H. C. King.

Disston Plantations.—J. H. Kraemer.

Palm Beach and Ormond.—Rev. J. N. MacGonigle, D.D.

St. Augustine.—Jno. N. MacGonigle, *Chairman;* Henry Gaillard, *Vice-Chairman;* Col. E. C. Bainbridge, Major Thos. H. Handbury, Capt. Henry Marcotte, Hon. Chas. Swayne, Hon. A. J. Corbett, J. K. Rainey, De Witt Webb, Dr. A. Anderson, J. S. Ingraham, W. W. Dewhurst, Geo. F. Miles, Josiah James, W. C. Stevens, C. B. Knott, R. T. Goff, Jno. T. Dismukes, H. A. Barling, Jr., C. F. Sperry.

Hotel-Headquarters.—At Ocala, the Ocala House; at Tampa, the Tampa Bay Hotel; at Palm Beach, the Royal Poinciana; at Ormond, the Hotel Ormond; at St. Augustine, the Ponce de Leon.

The second session of the meeting was held on Wednesday morning, March 27th, in the Ocala Opera House, where an address of welcome was delivered by Mayor John G. Reardon, and appropriate responses were made for the Institute by Dr. David T. Day and the Secretary. The hall was beautifully decorated for the occasion with flags, palm leaves and growing plants.

The third session was held at the Ocala House on Wednesday evening, March 27th, Dr. Day presiding, when the following papers were read and discussed:

The White Phosphates of Tennessee, by C. W. Hayes, Washington, D. C.

Geological Sketch of Florida, by E. T. Cox, Albion, Florida.

The Albion Phosphate District, by E. T. Cox, Albion, Florida.

The Florida Rock-Phosphate Deposits, by G. M. Wells, Ocala, Florida.

The fourth session was held at Tampa Bay Hotel, on Friday evening, March 29th, when the following papers were read and discussed:

Biographical Notice of Prof. Moritz F. Gaetzschmann, by R. W. Raymond, New York City.

The Florida Land- and Pebble-Phosphates, by E. W. Codington, Bartow, Florida.

Nickel and Nickel-Steel, by Francis L. Sperry, Cleveland, O. (In connection with this paper Mr. Sperry exhibited an interesting collection of samples, including articles manufactured from metallic nickel and its alloys.)

The following papers were presented in print:

The Present Limitations of the Cyanide Process, by C. W. Merrill, San Francisco, Cal.

Hysteromorphous Auriferous Deposits of the Tertiary and Cretaceous Periods in New Zealand, by Henry A. Gordon, Wellington, New Zealand.

Mining Leases, by Francis T. Freeland, Aspen, Colorado.

The Nomenclature of Zinc-Ores, by Walter Renton Ingalls, New York City.

A Water-Cooling Apparatus, by Carl Henrich, Ducktown, Tenn.

The Northeastern Bituminous Coal-Measures of the Appalachian System, by George S. Ramsay, McKeesport, Pa.

Cinnabar in Texas, by W. P. Blake, New Haven, Conn.

Further Experiments for Determining the Fusibility of Fire-Clays, by H. O. Hofman, Boston, Mass.

The Tin-Deposits of Durango, Mexico, by Walter Renton Ingalls, New York City.

North Carolina Monazite, by H. B. C. Nitze, Baltimore, Md.

Milling Arizona Gold-Ores with a "Colorado" Stamp-Mill, by Willard S. Morse, Prescott, Ariz.

Note on a Proposed Scheme for the Study of the Physics of Cast-Iron, by William R. Webster, Philadelphia, Pa.

Report of Assays of Copper and Copper Matte made in Accordance with the Plan Suggested by Dr. A. R. Ledoux in his paper on "A Uniform Method," etc., read at the Bridgeport Meeting.

A New Slag-Car for Lead and Copper Blast-Furnaces, by Carl Henrich, Ducktown, Tenn.

By a resolution, unanimously adopted, the Secretary was directed to express to Th. Grosswendt, Esq., Manager of the Hamburg Phosphate Company, Iverness, Florida, the sincere

disappointment and regret of the members of the Institute that the delay in returning from Silver Springs, on the 28th inst., rendered it impracticable for them to visit the mines of that company in accordance with their own desire and expectation, and with his hospitable invitation, for which the cordial thanks of the Institute are tendered.

The fifth and concluding session was held in the Casino, at St. Augustine, on Saturday evening, April 6th, when the report of the Scrutineers appointed at the New York session was presented, showing the following officers to have been elected:

PRESIDENT.

JOSEPH D. WEEKS,	Pittsburgh, Pa.

VICE-PRESIDENTS.

(To serve two years.)

WHEATON B. KUNHARDT,	New York City.
JAMES F. LEWIS,	Chicago, Ill.
CHARLES A. STETEFELDT,	Oakland, Cal.

MANAGERS.

(To serve three years.)

LEVI HOLBROOK,	New York City.
ALBERT R. LEDOUX,	New York City.
WILLIAM R. WEBSTER,	Philadelphia, Pa.

TREASURER.

THEODORE D. RAND,	Philadelphia, Pa.

SECRETARY.

ROSSITER W. RAYMOND,	New York City.

The following papers were read by title:

The Ducktown, Tennessee, Ore-Deposits, and the Treatment of the Ducktown Copper-Ores, by Carl Henrich, Ducktown, Tenn.

The Lixiviation of Silver-Ores by the Russell Process at Aspen, Colorado, by Willard S. Morse, Prescott, Ariz.

Notes on a Southern Coal-Washing Plant, by James J. Ormsbee, Tracy City, Tenn.

The Cyanide Process as Applied to Concentrates from a Nova Scotia Gold-Ore, by Richard W. Lodge, Boston, Mass.

The Treatment of Roasted Gold-Ores by Means of Bromine, by Richard W. Lodge, Boston, Mass.

The Equipment of Mining and Metallurgical Laboratories, by H. O. Hofman, Boston, Mass.

The Secretary announced the news, just received, of the appointment of Prof. T. M. Drown, an Honorary Member of the Institute and for many years its Secretary, to the presidency of Lehigh University, Bethlehem, Pa., and was instructed by unanimous vote to send to Prof. Drown the following telegram:

"The Institute in session heartily congratulates you upon an honor well deserved, and Lehigh University upon her wise choice of a new president."

After the unanimous adoption of a resolution instructing the Secretary to convey, by individual letters, to the individuals, firms, and corporations concerned, the thanks of the Institute for the abundant courtesy and hospitality extended to its visiting members and guests, the meeting was adjourned.

Members and Associates Elected.

The following persons were elected as members or associates at the sessions of the meeting:

Honorary Member.

Prof. Joseph Le Conte,	Berkeley, Cal.

Members.

Robert Hay Anderson,	Ponsonby, Mont.
Thomas Alexander Allen,	Melbourne, Aust.
Ernest F. Ayton,	Zacatecas, Mex.
Alan D. Bell,	Ely, Minn.
Russell M. Bennett,	Minneapolis, Minn.
C. C. Bolton,	Cleveland, O.
Lucius Polk Brown,	Nashville, Tenn.
William Clinton Brown,	Brooklyn, N. Y.
W. S. Carhart,	Rico, Colo.
Herman Bohn Christiansen,	Hermitage, Ga.
B. Dawson Coleman,	Lebanon, Pa.
Edward Coleman,	Lebanon, Pa.
Frank Hearne Crockard,	Benwood, W. Va.
Wayne Darlington,	Philadelphia, Pa.
Wythe Denby,	Morenci, Ariz.
Theodore Dengler,	Atlantic Mine, Mich.
Richard D. Divine,	Chicago, Ill.
Ernest C. Engelhardt,	Denver, Colo.
Charles Marvin Fassett,	Spokane, Wash.
Paul Revere Forbes,	Boston, Mass.

Henry G. Granger,	Paramaribo, Surinam.
John Rupert Joseph Gripper,	London, Eng.
Mellen Stanwood Harlow,	New York, N. Y.
D. Garth Hearne,	Wheeling, W. Va.
F. Hébert,	Paris, France.
Murray Innes,	Morenci, Ariz.
Matthew Buchan Jamieson,	Melbourne, Aust.
J. T. Jones,	Ocala, Fla.
B. B. Kann,	Pittsburgh, Pa.
H. W. Ferd. Kayser,	Waratah, Tasmania.
R. Brent Keyser,	Baltimore, Md.
G. A. Kornberg,	Butte, Mont.
J. B. Leggat,	Butte, Mont.
E. L. McGary,	Pittsburgh, Pa.
Charles H. McMahan,	Coahuila, Mex.
W. Clayton Miller,	Spokane, Wash.
Benjamin Franklin Morley,	Chester, Pa.
Prof. John Flesher Newson,	Bloomington, Ind.
J. C. Nichols,	Grand Junction, Colo.
Morris Brown Parker,	White Oaks, N. M.
Dalton Parmly,	Sharpsville, Pa.
William Manley Philipotes,	Marysville, Mont.
Heinrich Ries,	New York, N. Y.
Cyrus Robinson,	Columbus, O.
William Rotthoff,	Rankin, Pa.
George E. Somers,	Bridgeport, Conn.
Henry Souther,	Hartford, Conn.
George Steiger,	Washington, D. C.
Dirk Strumpel,	Guanajuato, Mex.
Gustave Thuillier,	Ocala, Fla.
William Francis Tindall,	Rapid City, So. Dak.
Daniel B. Waters,	Melbourne, Aust.
Rolla B. Watson,	Park City, Utah.
Hosea Webster,	New York, N. Y.
Arthur Weld,	Dahlonega, Ga.
Prof. Horace L. Wells,	New Haven, Conn.
Coryton M. Woodbury,	Middlesborough, Ky.
William Smith Yeates,	Atlanta, Ga.

Associates.

Adelbert D. Edwards,	Atlantic Mine, Mich.
Frank H. Hosford,	Washington, D. C.
Charles J. Hughes, Jr.,	Denver, Colo.
Arthur L. Nowell,	Boston, Mass.
Joseph Philips, Jr.,	Bethlehem, Pa.
P. A. Thomas,	Denver, Colo.
J. Fount Tillman,	Washington, D. C.

Associate Made Member.

Stanley Gifford,	Butte, Mont.

Excursions and Entertainments.

It is difficult to include, even in a bare catalogue, all the pleasures which were provided for the members and guests of the Institute by the generous hospitality of their hosts in all parts of Florida. Still less can any adequate description be conveyed by such a catalogue of the continuous and unalloyed delight with which this hospitality was enjoyed.

Among excursions, mention should first be made of the excursion which included all the rest, namely, the journey of the party from the north in special Pullman cars, which were retained for the whole period of more than two weeks. Without this provision of comfortable accommodations, it would have been impracticable to carry out the extended tour proposed; but apart from this feature of necessity, the arrangement, by keeping together as a unit a large company of congenial friends, enhanced immensely the pleasure and profit of the trip. From time to time the party was augmented by local members and hosts; and the opportunities thus given for professional as well as social intercourse, made the journey equivalent to many sessions and discussions. For the universally acceptable management of the complicated details of this tour, the Institute is indebted to the skill, energy, and patience of Dr. David T. Day and his assistant, Mr. Edward W. Parker.

Leaving Washington on Monday evening, March 25th, by the Southern Railway, the party stopped for several hours, on Tuesday, at Savannah, Ga., where excursions were made to "the Hermitage" and to the Bonaventure cemetery, and dinner was had at the De Soto Hotel.

On Wednesday, after the morning sessions at Ocala, an excursion was made to the Piedmont and Alachua phosphate-mines, the Blue Springs, and Homosassa, where a fish- and oyster-dinner was served. On the return to Ocala, the Dunnellon mines and works were inspected.

After the session of Wednesday evening, a delightful reception and entertainment was given by the Ocala Local Committee in the parlors of the hotel.

On Thursday the party proceeded by special train to Silver Springs, and enjoyed a trip on the beautiful Oklawaha river, including luncheon *al fresco* in the forest on its banks.

After the return to Ocala, the train left for Tampa Bay, arriving late on Thursday evening.

On Friday, March 29th, Port Tampa was visited by special train, and the harbor-dredge, the arrangements for shipping phosphates and other objects of interest were inspected.

On Saturday a delightful excursion was made by special train to Port Tampa, and by steamer on the bay to the quarantine station on Mullet Key, the lighthouse and winter gardens on Egmont Key, and several beautiful private estates on the coast. In the evening a complimentary hop was given at the hotel to visiting members and ladies.

On Sunday, March 31st, services in the churches of Tampa were attended, and a special service was held in the evening at the hotel.

On Monday the train proceeded to Bone Valley, visiting the land-pebble phosphate-mine of the Bone Valley Company and the new plant of the Palmetto Phosphate Company, and thence to Kissimmee and Runnymede, where luncheon was served by the Disston Land Company, and the extensive operations of the company were inspected with interest.

For the free transportation of special trains over the lines of the Plant System in Florida, and for the excursions from Tampa Bay, acknowledgments are due to Mr. H. B. Plant, and also to Col. B. W. Wrenn and the other attentive and courteous officials of the Plant System.

Travelling through the night, *via* Sanford and Titusville, the party arrived on Tuesday morning at Palm Beach, Lake Worth, and was received into the Royal Poinciana hotel. Visits to the beach, surf-bathing, walks along the "trails" through the groves of sub-tropical trees, and a sail on Lake Worth northward occupied the day. In the evening an informal reception was held in the hotel, at which Rev. J. N. MacGonigle, as the representative of Mr. H. M. Flagler, extended to the Institute a cordial welcome to the East Coast of Florida.

On Wednesday, April 3d, by the courtesy of Mr. Flagler, an excursion was made by steamer on Lake Worth southward nearly to the mouth of the canal now under construction for connection with Biscayne Bay. Luncheon was served on board the steamer. The evening was pleasantly occupied with music, conversation and an amusing "mock trial" improvised by

members of the party and held before Dr. MacGonigle as judge and a jury of ladies.

On Thursday morning, April 4th, the special train proceeded along the lovely Indian river to Ormond, where the afternoon was spent in walking or driving along the magnificent Ormond Beach or bathing in the surf, and the evening in conversation and music.

Friday, April 5th, was occupied in an excursion to "the Cabin," a picnic station in the forest, to which half the party were conveyed on the Halifax and Tomoka rivers in steam-launches, while the other half were driven in carriages, the two sections exchanging modes of transportation for the return. It was found impossible to decide afterwards which route was the more delightful, and the statement of one of the genial managers of the hotel, that "the drive was almost better than the sail," was unanimously adopted as a compromise. The distance by land being relatively short, the drive was extended to include a visit to the lovely town of Daytona, with its picturesque cottages and wonderful avenue shaded by live-oaks draped in pendant moss.

On Saturday morning, April 6th, the party left for St. Augustine, where the afternoon was spent, under the guidance of the local committee, in sailing on the Matanzas river and inlet to North Beach, visiting Anastasia Island, driving to the old Spanish fort and through the picturesque narrow streets of "the Ancient City," and inspecting Dr. Vedder's quaint and curious museum of the natural history of Florida. A delightful reception was given at "The Manse" by Dr. and Mrs. MacGonigle, to which the band of the 3d Artillery, U. S. A., contributed the additional charm of excellent music. In the evening, after the closing session of the Institute, a social reception was given to the Institute in the Casino, by C. B. Knott, Esq., manager of the Ponce de Leon.

On Sunday service was attended, morning and afternoon, in the magnificent Flagler Memorial Church, of which Dr. MacGonigle is pastor, and, later in the afternoon, the members and guests of the Institute were cordially received by Mr. Franklin Smith in his beautiful Moorish "Villa Zorayda" (reproducing, in many respects, the architecture and decorative details of the Alhambra), where their pleasure in the inspection

of numerous treasures of art was enhanced by the interesting explanations and comments of their hospitable host.

On Monday morning, after a final pleasant experience as the guests of Colonel Edmund C. Bainbridge, 3d Artillery, U. S. A., at morning parade, the party left St. Augustine for the north.

A highly-appreciated feature of the return trip was the fast run of the special train over the Southern Railway, from Columbia, S. C., to Washington, which was made in two hours less than the schedule-time of the "Florida Limited," in order to make connections at Washington which practically saved a night's travel to northern members.

Members, Associates and Guests Registered.

No complete registry was kept of those participating in the various sessions and excursions of the meeting. The following list, doubtless, fails to include many local members and guests who did not accompany the party throughout, or did not register at hotel-headquarters. The traveling party included a large proportion of ladies, amounting at one time to more than fifty in number.

James Archbald.
W. Clinton Brown.
Jaudon Brown.
J. P. Carson.
H. S. Chamberlain.
George H. Clapp.
C. S. Clarke.
Lee Clymer.
E. W. Codington.
W. B. Cogswell.
Edgar S. Cook.
David T. Day.
Thomas S. Disston.
B. F. Fackenthal, Jr.
John S. Fackenthal.
B. E. Fernow.
H. R. Hall.
Edward P. Hamilton.
A. Hardt.
C. H. Hitchcock.
Thomas Hoatson.
Thomas Hoatson, Jr.
Edward Holder.
Frank H. Hosford.
J. E. Johnson.
J. T. Jones.
Washington Jones.
Paul S. King.
S. H. Knight.
Edward K. Landis.
P. D. Langdon.
J. H. Lee.
V. C. McCormick.
J. N. MacGonigle.
J. G. McIlvain.
W. R. McIlvain.
Otto A. Moses.
George Ormrod.
E. W. Parker.
W. W. Pickford.
R. W. Raymond.
W. M. Rexford.
M. G. Riché.
Howard Riegel.
G. S. Saylor.
J. F. Saeger.
H. J. Seaman.
A. W. Sheafer.

William L. Sheafer.
F. L. Sperry.
E. J. Spindle.
H. R. Stanford.
Edwin Thomas.
W. E. Thomas.
G. Thuillier.
E. B. Toedt.
H. G. Torrey.
Gray Torrey.
Howard Valentine.
M. D. Valentine.
Willard Warner, Jr.
S. Whinery.
Charles Wiley.
W. H. Wiley.
John Wilkes.
Oliver Williams.

Proceedings of the Sixty-Ninth Meeting, Atlanta, Georgia, October, 1895.

Local Committee.—Prof. W. S Yeates, *Chairman;* Prof. W. H. Emerson, *Secretary;* Messrs. G. C. Hewett, William M. Brewer, James A .Burns.

Citizens' Committee.—Hon. Hoke Smith, *Chairman;* Hon. Clarke Howell, Hon. W. A. Hemphill, Hon. W. A. Wright, Hon. W. H. Venable, Hon. R. B. Bullock, Col. R. F. Maddox, Messrs. C. A. Collier, H. H. Cabaniss, Forrest Adair, John W. Grant, E. P. Chamberlin, F. P. Rice, R. D. Spalding, S. W. Goode, M. F. Amorous, C. E. Harmon, R. P. Beecher.

Excursions to Stone Mountain and Marble Hill.—W. M. Brewer, G. C. Hewett.

Excursion to Birmingham.—Prof. William B. Phillips, Charles P. Perin.

Reception at Lookout Mountain.—Captain H. S. Chamberlain, Chattanooga, Tenn.

In Charge of Special Train.—E. W. Parker, Washington, D. C.

Hotel Headquarters.—Hotel Aragon, Atlanta, Ga.

The first session was held in the hall of the Concordia Association, on Tuesday evening, October 8th. Addresses of welcome were made by Prof. W. S. Yeates, State Geologist of Georgia, and Chairman of the Local Committee; Mr. Alexander W. Smith, General Manager of the Cotton States and International Exposition, and Dr. I. S. Hopkins, President of the Georgia School of Technology. President Joseph D. Weeks, after responding in behalf of the Institute, proceeded to deliver the Presidential Address (to be published separately.)

The Secretary presented, with a few remarks, two biographical notices of Prof. Franz Posepny and Hon. Eckley B. Coxe respectively, which were distributed in pamphlet form.

The second session was held at the Hotel Aragon on Wednesday morning, October 9th.

The Secretary gave notice of the following proposed amendments to the rules, to be considered at the next annual meeting.

Rule II.—It is proposed to amend this rule by adding the following: "and members or associates whose post-office addresses are outside of the United States, Canada and Mexico, shall not be entitled to vote by mail except upon proposed amendments to the Rules."

The reason for this proposed amendment was stated to be that, although the Institute is an American society, the increasing number of foreign members neces-

sitates, according to the present system, the sending of many ballots to places so remote that they cannot be returned in time to be counted under the rule. The privilege of voting is thus practically nullified as to members abroad; and it is believed, moreover, that the policy of confining the privilege to the membership in the three countries named would be acceptable to all, as not only saving useless expense, but placing the control of the Institute as to the election of officers, etc., formally (as it is now substantially) in the hands of resident American members and associates. It will be noticed that the proposed amendment does not affect the right of any member or associate, wherever resident, to vote in person at the meetings of the Institute.

Rule III.—It is proposed to amend this rule by striking out the words "at the annual meeting," and substituting the words "on the first day of each calendar year."

The reason of this proposed amendment is stated to be, that the present rule fixes a somewhat variable date, and complicates in various ways the keeping of the accounts. Thus the financial statement of the Council, presented at the annual meeting in February, is necessarily closed January 31st, at which date there are still to come several weeks of collections and expenditures, which properly belong to the current year of the Institute. Under the proposed change, the accounts closed December 31, duly audited, and presented at the annual meeting in February, could be more conveniently kept and would more accurately exhibit the financial condition of the Institute. The dues are now payable in advance at the February meeting, but are in fact collected by mail, through bills sent to members after that date. Under the proposed amendment, these bills would be sent out in January; but the liberal allowance of several months for delay in payment made by the Council would cause this change to be practically unfelt by members not in arrears for more than one year.

Rule VI.—It is proposed to amend this rule by striking out the provision for special meetings, and changing the second sentence of the rule so that it shall read, "Other meetings shall be held at such times and places as the Council shall select, and notice of all meetings shall be given," etc.

The reason for this proposed amendment is stated to be, that the provision requiring two meetings in each year besides the annual meeting is not advisable, and has been repeatedly ignored for good cause by the Council. It is believed that this matter should be explicitly left (where it already practically rests) in the discretion of the Council.

The provision that special meetings shall be called at the request of fifteen members is manifestly improper in view of the present large membership of the Institute. The good sense of members has hitherto ignored it; and it is believed to be entirely unnecessary.

Rule VII.—It is proposed to amend this rule by making the title "Papers and Publications," and by inserting after the first paragraph the following: "The published papers and volumes

of Transactions shall be distributed to all members and associates not in arrears, and may be sold to the public, upon such conditions as the Council shall prescribe; but the Council may in its discretion omit sending to members and associates outside of the United States, Canada and Mexico, special circulars, unless the same contain proposed amendments to the rules."

Also, by adding to the rule the following: "Nor shall the Council or the Institute officially approve or disapprove any technical or scientific opinion, or any proposed undertaking outside the management of the meetings, discussions, and publications of the Institute as provided in these Rules."

The reasons for these proposed amendments are stated to be:

1. Many special circulars, such as the final programme of meetings about to be held, are not calculated to interest foreign members, and would reach them too late to be of use to them. It is believed that the sending of such circulars may safely be left to the discretion of the Council.

2. The proposed final addition to the rule is intended to express, formally, what has been the uniform practice of the Council and the Institute for many years. The large, scattered and varied membership of the Institute, and the comparatively small proportion of it attending any one meeting, have long been recognized as precluding the propriety of action committing the whole body to any proposition, however meritorious. And it has been similarly recognized, that what the Institute could not properly do, the Council ought not to do in its name. The formal prohibition expressed in the proposed amendment would not alter in any way the existing practice, but would constitute a convenient notice and explanation in many cases in which the influence of the Institute is besought in aid of professional, national, and international reforms and enterprises, which, however excellent in themselves, are outside of its legitimate purposes.

The Secretary announced that the foregoing proposed amendments had been approved by the Council for consideration at the annual meeting, at which time they would be open to discussion and amendment, before their final submission to the members for ballot by mail.

The following papers were then presented and discussed:

The Present Condition of Gold-Mining in the Southern Appalachian States, by H. B. C. Nitze and H. A. J. Wilkens, Baltimore, Md.

The Gold-Regions of Georgia and Alabama, by William M. Brewer, Atlanta, Ga.

Notes and Recollections Concerning the Mineral Resources of Northern Georgia and Western North Carolina, by W. P. Blake, New Haven, Conn.

The Assay of Auriferous Ores and Gravels by Amalgamation and the Blow-pipe, by R. W. Leonard, Kingston, Ontario, Can.

Mining Titles on Spanish Grants in the United States, by R. W. Raymond, New York City.

The third and final session was held at the Hotel Aragon on Wednesday afternoon, October 9th.

The following papers were read and discussed:

The Magnetization and Concentration of Iron-Ores, by William B. Phillips, Birmingham, Ala.

Southern Magnetites and Magnetic Separation, by Harvey S. Chase, NewYork City.

The Magnetic Separation of Iron-Ore, by C. M. Ball, Saratoga Springs, N. Y.

Specifications for Steel Rails of Heavy Sections Manufactured West of the Alleghenies, by R. W. Hunt, Chicago, Ill.

Notes on the Kaolin and Clay Deposits in North Carolina, by J. A. Holmes, State Geologist, Chapel Hill, N. C.

Notes on the Underground Supplies of Potable Waters in the South Atlantic Piedmont Plateau, by J. A. Holmes, State Geologist, Chapel Hill, N. C.

The following papers were presented by the Secretary in print or manuscript, with oral abstract of their contents, the authors being absent:

The Monazite Districts of North and South Carolina, by C. A. Mezger, Shelby, N. C.

A Section of Rich Patch Mountain at Iron Gate, Va., by E. J. Schmitz, New York City.

The Phosphates and Marls of Alabama, by Eugene A. Smith, State Geologist, University, Ala.

Chrome in the Southern Appalachian Region, by William Glenn, Baltimore, Md.

The Eastern Coal-Regions of Kentucky, by Graham Macfarlane, Louisville, Ky.

Onyx-Marbles, by Courtney DeKalb, Rolla, Mo.

Folds and Faults in Pennsylvania Anthracite Beds, by Benjamin Smith Lyman, Philadelphia, Pa.

The Geological Structure of the Western part of the Ver-

milion Range, Minn., by H. L. Smyth, Cambridge, Mass., and J. R. Finlay, Virginia, Minn.

A Comparison of Recent Phosphorus-Determinations in Steel, by George E. Thackray, Johnstown, Pa.

The Determination of Graphite in Pig-Iron, by Porter W. Shimer, Easton, Pa.

The Assay of Silver-Sulphides, by Howard Van F. Furman, Denver, Colo.

The Effect of Washing with Water upon the Silver Chloride in Roasted Ore, by Willard S. Morse, Prescott, Ariz.

The Theory and Practice of Ore-Sampling, by D. W. Brunton, Aspen, Colo.

The Form of Fissure-Walls, as Affected by Sub-Fissuring and by the Flow of Rocks, by William Glenn, Baltimore, Md.

Notes on Certain Water-Worn Vein-Specimens, by F. C. Holman, San Francisco, Cal.

The following papers were read by title:

Stamp-Milling in the Black Hills, South Dakota, and at Grass Valley, California, by T. A. Rickard, Denver, Colo.

An Improved Form of Protractor for Mapping Mine-Surveys, by W. S. Ayres, Hazleton, Pa.

Corundum of the Appalachian Crystalline Belt, by J. V. Lewis, Assistant Geologist, Chapel Hill, N. C.

After the unanimous adoption of a resolution directing the official expression of the thanks of the Institute for courtesies received, the meeting was adjourned.

Members and Associates Elected.

The following persons were elected as members or associates at the sessions of the meeting:

Members.

Thomas Johnson Britten, ...	Johannesburg, S. A. R.
Horace F. Brown, ...	Chicago, Ill.
Raymond B. Brown, ...	Allegheny, Pa.
George H. Clark, ...	Cedartown, Ga.
Walter Nathan Crafts, ...	Troy, N. Y.
David Evans, ...	Middlesborough, England.
J. Ralph Finlay, ...	Virginia, Minn.
B. M. Hall, ...	Atlanta, Ga.
Frederic Hall Harvey, ...	Galt, Cal.
George H. Hooper, ...	Hague, N. Y.
Albert Huessener, ...	Gelsenkirchen, Germany.

William Spencer Hutchinson,	Galt, Cal.
Joseph Esrey Johnson, Jr.,	Longdale, Va.
Charles B. Kingston,	Aspen, Colo.
A. B. Richfield,	Rico, Colo.
Harry Huntington Miller,	New York City.
Charles L. Patterson,	Chicago, Ill.
Robert Swain Perry,	Cave Springs, Ga.
John B. Piggott,	Bessemer, Mich.
Frederick Powell,	Charlotte, N. C.
N. P. Pratt,	Atlanta, Ga.
Charles Henry Quereau,	Plattsmouth, Neb.
F. H. Seamon,	Matehuala, Mex.
C. M. Shanahan,	Low Moor, Va.
Robert B. Turner,	Butte, Mont.
Robert Chester Turner,	Bodie, Cal.
Samuel Herbert Williams,	Butte, Mont.

ASSOCIATES.

W. L. Kann,	Pittsburgh, Pa.
Thomas G. McKell,	Chillicothe,
Randolph Stalnaker,	Wheeling, W. Va.
Joel F. Vaile,	Denver, Colo.

The following persons were elected by mail, June, 1895.

MEMBERS.

Thomas Armstrong,	Harqua Hala, Ariz.
Henry R. Batcheller,	Norris, Mont.
Charles S. Clarke,	Ocala, Fla.
A. L. Collins,	Central City, Colo.
John Dickson Cosens,	Oorgaum, So. India.
Edward Wilkins Dewey,	New York, N. Y.
Thomas R. Ellesbeck,	Salt Lake City, Utah.
Thomas G. Greenway,	N. Adelaide, So. Australia.
Edward Hiller,	Ocala, Fla.
J. A. Holmes,	Chapel Hill, N. C.
Joseph Volney Lewis,	Chapel Hill, N. C.
Harris K. Masters,	Brooklyn, N. Y.
Edmund Howd Miller,	New York, N. Y.
Prof. Charles E. Munroe,	Washington, D. C.
William Plummer,	De Lamar, Idaho.
A. L. Read,	Lead, So. Dakota.
Ernest H. Simonds,	Berkeley, Cal.
Edward Skewes,	Cripple Creek, Colo.
P. Bosworth Smith,	Oorgaum, So. India.
William Alfred Tucker,	Boston, Mass.
William Young Westervelt,	Isabella, Tenn.
Ashley Hope Wynne,	Sinaloa, Mexico.

ASSOCIATES.

R. L. Armit,	Colorado Springs, Colo.
Benjamin Palmer Carter,	Sheffield, Ala.
Charles S. Herzig,	New York, N. Y.
Richard S. McCaffery,	New York, N. Y.
Richard de Bellevue Smith,	Anaconda, Mont.

Associates made Members.

John Roderick McKay,	Lucknow, N. S. W.
George A. Waller,	Glenlusk, Tasmania.

The following is the list of those elected by mail, August, 1895:

Members.

Ai Arthur Abbott,	Coulterville, Cal.
C. O. Baker,	Newark, N. J.
Baron René de Batz,	Paris, France.
Samuel Edward Bowlby,	Marysville, Mont.
Owen Byrnes,	Granite, Mont.
Charles Catlett,	Staunton, Va.
J. A. Chalmers.	Johannesburg, S. A. R.
William Coumerilh,	Murray, Idaho.
J. R. Farrell,	Baker City, Ore.
Fred. Nathaniel Fletcher,	Helena, Mont.
Ware B. Gay,	Richmond, Va.
William Frank Grace,	Charters Towers, Queensland.
Andre P. Griffiths,	London, England.
Preston Hampton Haskell,	Oxmoor, Ala.
William Henry Howard,	Pueblo, Col.
Edward Kellar,	Baltimore, Md.
James Herbert Kervin,	Phillipsburg, Mont.
Anthony Francis Lucas,	Avery Island, La.
Thomas Morcom,	Baker City, Ore.
Jonas J. Pierce,	Sharpsville, Pa.
William Duncan Sawers,	Glasgow, Scotland.
William G. P. Sharp,	Coolgardie, W. Aust.
Thomas Starbird,	Cornucopia, Ore.
George Thomas, 3d,	Burnham, Pa.
Dudley A. Van Ingen,	Brooklyn, N. Y.
George B. Wardman,	Salt Lake City, Utah.
W. d'H. Washington,	New York, N. Y.
Frank Western,	London, England.
Hamilton M. Wingate,	Cassilis, Aust.
Clarence Ashley Woodford,	Buluwayo, Metabeleland, S. A. R.
J. Wilson Woodrow,	Sierra Mojada, Mex.
James Smellie Young,	Glasgow, Scotland.

Associates.

James Miller,	Nogales, Ariz.
W. S. Nelson,	Denver, Col.
Robert Scott Weir,	Nogales, Ariz.

Associates made Members.

Benjamin Palmer Carter,	Sheffield, Ala.
A. D. Edwards,	Atlantic Mine, Mich.
Charles S. Herzig,	New York, N. Y.

EXCURSIONS AND ENTERTAINMENTS.

By the courtesy of the Southern Railway, a special excursion-train of Pullman cars, conveying members and guests of the Institute, was hauled free from Washington to Atlanta, and back to Washington *via* Chattanooga and Asheville.

The Cotton States and International Exposition at Atlanta was visited on Tuesday afternoon, October 8th, on Wednesday evening (when a magnificent display of fire-works was given, including a special piece representing the seal of the Institute), and during the whole of Thursday.

On Thursday evening a reception was given by Dr. David T. Day and Hon. B. E. Fernow, chiefs of the departments of minerals and forestry at the Exposition, assisted by Mrs. Day and Mrs. Fernow, in the Minerals and Forestry Building. The inspection of the admirable exhibits in these departments, and a delightful series of musical entertainments and recitations, concluding with a supper, held the large company to a late hour.

On Friday, a special train conveyed the party (amounting to nearly two hundred), to the famous granite-quarries at Stone Mountain, where the guests were hospitably received by the Messrs. Venable, the proprietors, and the ladies of their household, and, after inspecting the quarries and the striking scenery of the place, enjoyed the unique entertainment of a barbecue in the grove near the residence of their hosts.

A party of about twenty-five left Atlanta Friday night, to spend Saturday in visiting mines and works in the Birmingham district, Alabama, rejoining the main body of excursionists Sunday morning at Chattanooga. Through the courtesy of the Louisville and Nashville Railroad Co., this party was provided with transportation to the Ishkooda (formerly Eureka) mines of the Tennessee Coal, Iron and Railway Co., now operated by J. W. Worthington & Co., where opportunity was given to observe the mining of soft and hard red ore; and thence to Bessemer, where a brief visit was made to the experimental magnetizing- and concentrating-works, and a longer one to the pipe-works and new coke-ovens of the Howard-Harrison Iron Co. These works use the system of Mr. Erskine Ramsay for utilizing the waste heat from the bee-hive ovens to raise steam, and heat the core-ovens. The coke is made from Blue Creek slack-coal, washed in the Robinson washer, and is all 72-hour coke.

During the morning, stops were made at Ensley and at the Sloss furnaces; and in the afternoon some of the party visited the furnaces of the Pioneer Company at Thomas and the Robinson coal-washer at Pratt Mines.

On Saturday, a most picturesque and interesting excursion was made by special train on the Marietta and North Georgia railroad to the marble quarries of the Southern Marble Company and the Georgia Marble Company at Marble Hill, where the party was hospitably entertained and greatly interested.

Saturday night, the special train left Atlanta, arriving at Chattanooga early Sunday morning, when the party proceeded to Lookout Inn, on Lookout Mountain, where the day was spent. A special religious service was held in the evening.

Monday, October 14, was devoted to an excursion in carriages through the new National Park, covering the battle-field of Chickamauga, and already adorned with numerous interesting monuments of that battle.

On Monday evening a reception was held at Lookout Inn, after which, at a late hour, the party descended to Chattanooga and resumed its special train for the homeward journey.

Breakfast was had at Hot Springs, N. C., and Asheville was reached at about 9 A.M. After dinner at the Battery Park Hotel, carriages were in waiting to convey the party to leading points of beauty and interest, and finally through the beautiful grounds of Mr. Vanderbilt to his magnificent new mansion, now in process of completion. The superb autumn weather, which had prevailed throughout the week, reached its climax on this day; and the always lovely scenery of Asheville received a supreme charm from the hues of sky and forest.

Leaving Asheville at 4 P.M., the special train reached Washington on the following morning, and with many congratulations and regrets, the congenial fellow-travellers bade each other farewell, to scatter on their several ways.

Members, Associates and Guests Registered.

The following list comprises the names registered at the official headquarters in the Hotel Aragon, Atlanta. It is doubtless incomplete as a catalogue of those who at one time or another participated in the sessions and excursions of the meeting. A large number of ladies—not less than fifty—

was present, and greatly enhanced the social pleasure of the meeting.

William H. Adams.
James Archbald.
Henry Belin, Jr.
H. P. Bellinger.
L. C. Bierwirth.
George Herric Billings.
R. M. Blankenship.
Clement Le Boutillier.
William M. Bowron.
C. R. Boyd.
W. H. Bradley.
William M. Brewer.
Lucius P. Brown.
J. Blodget Britton.
Charles Carroll.
William H. Case.
H. S. Chamberlain.
Harvey S. Chase.
H. B. Christiansen.
George Huntington Clark.
Dr. E. W. Closson.
F. K. Copeland.
Torbert Coryell.
W. M. Courtis.
W. R. Crandall.
H. Dailey.
W. F. Downs.
David Evans.
B. E. Fernow.
H. S. Fleming.
J. W. Fuller.
J. W. Fuller, Jr.
John M. Garvin.
B. M. Hall.
C. W. Hayes.
G. C. Hewett.
Joseph T. Hilles.
L. Holbrook.
J. A. Holmes.
W. S. Hungerford.
R. W. Hunt.
J. E. Johnson, Jr.
E. S. Jones.
Washington Jones.
W. J. Keep.
William Kent.
Charles Kirchhoff.
J. H. Lee.
J. F. Lewis.
J. V. Lewis.
John Lilly.
William Lilly.
J. Lodge.
J. C. Lynes.
S. W. McCallie.
John M. McCandless.
Charles McCrery.
H. McCormick, Jr.
Graham Macfarlane.
H. H. Miller.
S. F. Morris.
H. B. C. Nitze.
Paul A. Oliver.
G. S. Page.
William G. Parke.
E. W. Parker.
Leonard Peckitt.
C. P. Perin.
R. S. Perry.
William B. Phillips.
F. E. Platt.
Jos. C. Platt.
S. M. Pitman.
Theo. D. Rand.
R. W. Raymond.
W. M. Rexford.
Ellen H. Richards.
J. S. Robeson.
Horace See.
A. W. Sheafer.
W. L. Sheafer.
A. M. Shook.
E. M. Shoup.
C. D. Simpson.
J. William Smith.
Oberlin Smith.
William T. Smith.
Adolph Thies.
Gray Torrey.
H. G. Torrey.
John P. Wanner.
J. D. Weeks.
W. Y. Westervelt.
E. L. Wiles.
H. A. J. Wilkens.
H. M. Wilson.
W. S. Yeates.

PAPERS.

Further Experiments for Determining the Fusibility of Fire-Clays.

BY H. O. HOFMAN, BOSTON, MASS.

(Florida Meeting, March, 1895.)

A PREVIOUS paper* has described the experiments made by Mr. C. D. Demond and the present writer to ascertain by an indirect method the refractory values of clays. Fire-clays were mixed with varying proportions of calcium carbonate and calcium carbonate and silica, to render them fusible at temperatures below the melting-point of platinum, and common brick-clays with alumina and silica to decrease their fusibility; the object being to arrive at a standard temperature at which fire-clays as well as common brick-clays could be tested; the amount of ingredient required by each clay for fusion being the measure of its refractoriness. The behavior of the samples in the fire gave such a satisfactory series, both in the descending scale with fire-clay and the ascending scale with common brick-clay, that it seemed an easy matter to assume a standard temperature of 1500° C., and to add fluxing or refractory substances to the clays until they melted at this temperature. This was found, however, to be very difficult. Samples of Mount Savage clay with from 40 to 65 per cent. of calcium carbonate and mixtures of calcium carbonate and silica, showed very plainly the effect of the successive additions, but there was only a gradual transition instead of the complete change which had been anticipated; the most refractory cone remaining erect or nearly so, the next bending slightly, and so on by degrees, until, in the last of the series, the apex touched the base. It is possible to arrive at somewhat satisfactory results by assuming the behavior of a certain cone within a given time as a standard and working out from this; but such a method

* *Trans.*, xxiv., 42.

becomes necessarily individual and can scarcely be made applicable for general use.

Seger's Direct Method.

The direct method of testing is to place a sample of clay, with accepted standards for comparison, in a crucible, and expose them in a suitable furnace to different temperatures until the sample shows the same degree of fusion as one of the standards. The seven clays of Bischof,* which formed the standards for a number of years, not being any longer obtainable (the character of the deposits having changed), Seger prepared graded mixtures of potash, lime, alumina and silica to take the place of the Bischof clays.

The following is a description of the experiments carried out in the metallurgical laboratory of the Massachusetts Institute of Technology with some of the leading fire-clays of this country according to Seger's method.† Acknowledgments are due to Mr. G. H. Anderson, S. B., and to Mr. R. Loring, S. B., for their careful and patient work which made it possible to carry to a successful end the large number of experiments necessary. The apparatus, etc., was all imported from the *Chemisches Laboratorium für Thonindustrie*, at Berlin, to insure accuracy in the work.

1. *The Furnace.*—The furnace used was the Deville furnace, represented in Figs. 1, 2 and 3. This is a small cylindrical furnace, *a*, of $\frac{1}{8}$-inch sheet-iron, lined with refractory material. It is open at the top and closed near the bottom by a cast-iron plate, *b*, having a large central opening surrounded by three rows of small perforations. Below this is the air-chamber, *c*, with blast inlet-pipe, *d*. The furnace rests upon the iron plate, *e*, which has an upturned edge and three legs, which are riveted to it. The place where furnace and plate meet is luted with a sandy non-shrinking clay. The furnace is lined for the first $5\frac{3}{8}$ inches with Spaeter's sintered magnesite from Veitsch, Styria, the rest of the lining being a mixture of 90 per cent. magnesite and 10 per cent. Zettlitz kaolin. The composition of the two refractories is shown by the following analyses:

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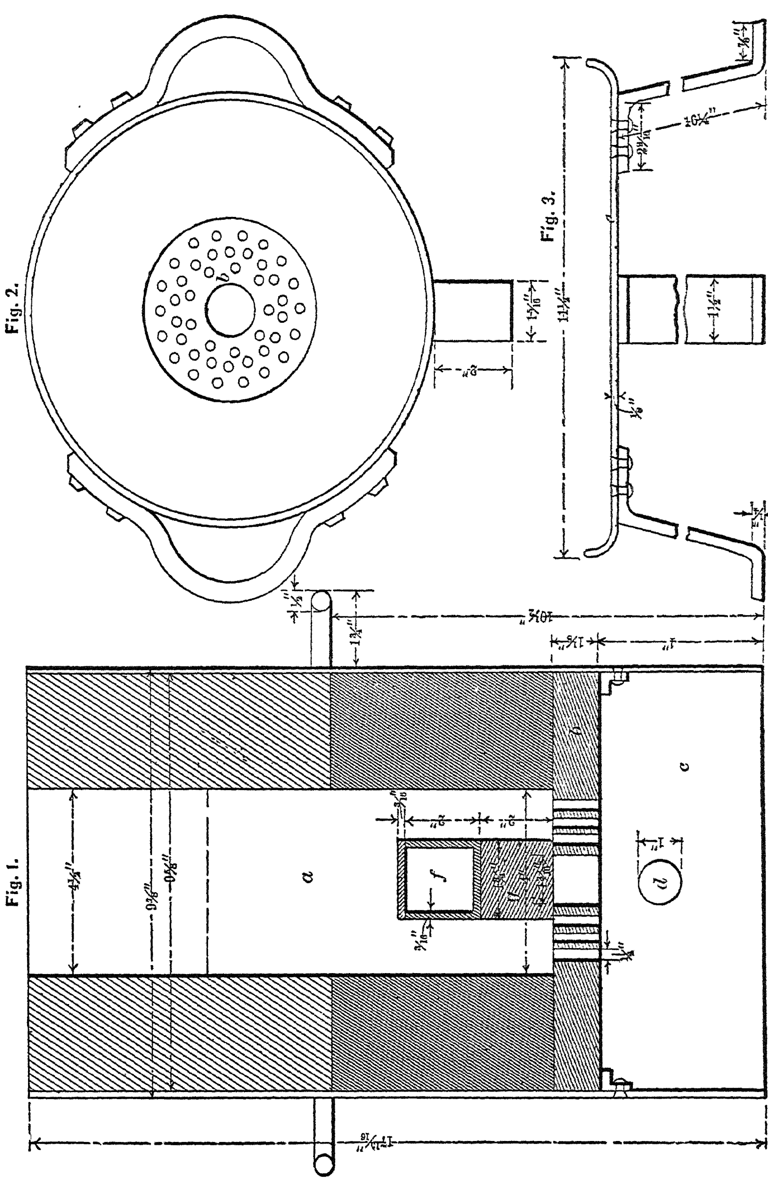
* *Trans.*, xxiv., 45, 46.

† *Thonindustrie Zeitung*, 1893, 1281.



DEVILLE FURNACE.

TABLE I.—*Analyses of Refractory Materials.*

	Al_2O_3.	SiO_2.	MgO.	CaO.	K_2O.	Fe_2O_3.	Chemist.
Sintered magnesite*......	.86	2.35	88.22	0.87	....	7.07	Winkler.
Zettlitz kaolin†............	38.54	45.68	0.38	0.08	0.66	0.90	Bischof.

After carrying out about sixty fusions the lower part of the lining began to be affected. This was probably caused by the ash of the charcoal and gas-carbon. The slag spread somewhat over the small perforations in the grate and had to be pried off. Some of the unaltered lining adhered to the slag and broke off, thus enlarging the cylinder and making it require more fuel. The furnace-lining cannot well be patched, and so it must be renewed. For this purpose the lining is broken out, crushed to pass a 4-mesh sieve (the scorified part being also used), moistened with from 8 to 10 per cent. of hot water, covered and kept in a sheltered place for from two to three days, that the water may be thoroughly absorbed. The hydration of magnesia causes the mass to harden slightly, when it is crushed again, if necessary, and moistened with sufficient water to make it cohere into a lump when squeezed in the hand, but not to wet the hand. The core around which the lining is to be tamped is a wooden truncated cone, $17\frac{3}{4}$ inches high and $3\frac{1}{2}$ and 4 inches in diameter, having a spindle $1\frac{1}{8}$ inches in diameter at the smaller end, which fits into the central opening of the cast-iron grate. It is enveloped in five layers of newspaper, the last one being fastened with mucilage to prevent its unrolling. When the core is in place the lining is filled in in layers about 2 inches thick, rammed down with a stick, say $\frac{3}{4}$ inch in diameter, new magnesite, prepared in the same way as the old, being used against the core, old magnesite against the iron shell for a depth of 6 inches, after which the old is exclusively used, the top layer being tamped so as to rise toward the core and pounded down firmly with an iron mallet or hammer. The furnace is now laid on its side, and the core, loosened by gently tapping the spindle, is drawn out. Most of the paper will adhere to the lining, to be burnt out later on. It is necessary to remove the core as soon as the

* Circular of the Pittsburgh Testing Laboratory. † *Trans.*, xxiv., 45.

tamping is finished, as it is liable to swell, thus causing difficulty in getting it out and loosening the lining. Should this happen, it is best to take it all out and reline the furnace at once, instead of having to do it a little later. The furnace is now placed on two bricks and air-dried for from 5 to 6 days, and the remaining moisture is finally driven off by keeping a wood-fire going in it for two days, no firing being necessary during the intervening night.

In experimenting with the furnace, one improvement in the construction suggested itself. After an experiment, the gas-carbon surrounding the support and part of the crucible is discharged through the central opening in the grate into the air-chamber, to clear which the furnace has to be lifted, which necessitates fresh luting. This might be avoided by having an opening with collar in the supporting plate to be closed by a hinged door and fastened with a tightening screw, an asbestos packing in the mortice of the door serving to make the joint air-tight.

The top of the furnace is uncovered. To draw off the gases, ashes, and particles of finely-divided fuel into the flue, a piece of sheet-iron with peep-hole was suspended in an inclined position over the furnace. In this way it was an easy matter to watch the surface of the charge, remove the crucible as soon as it became visible, and clean the furnace when an experiment was completed.

2. *The Crucible and its Support.*—The crucible, *f*, in which the clay and standard cones are placed to be brought to the intense heat necessary for fusion, consists of equal parts of calcined alumina and Zettlitz kaolin, mixed with sufficient raw kaolin to permit the whole to be properly moulded. It is so infusible that the most refractory clay, the Rokonitz clay-slate of Saarau, Silesia, can be melted down in it without its being appreciably injured by the high temperature. The lid is of the same material as the crucible. The crucible formerly used,* consisting of a mixture of calcined magnesite and chromite, moistened with magnesium chloride, and poured with alumina, has been given up, since the lining, as well as the chromic oxide, somewhat affected the samples that were being tested.

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* *Thonindustrie Zeitung*, 1892, p. 676; 1893, p. 1281.

The crucible support, *g*, is made of refractory material which begins to show signs of fusion only at a temperature represented by Seger Cone No. 35. It used to be made with a small cylindrical projection at the bottom, which fitted into the central opening of the grate, *b*, and thus insured its being directly in the middle. This was very convenient when starting work in a cold furnace where the grate is invisible.

3. *Seger Cones and Mould.*—The composition of the artificial refractory mixtures prepared by the late Prof. Seger to take the place of the standard fire-clays of Dr. Bischof is shown in Table II:

TABLE II.—*Seger Cones Compared with Bischof's Standard Clays.*

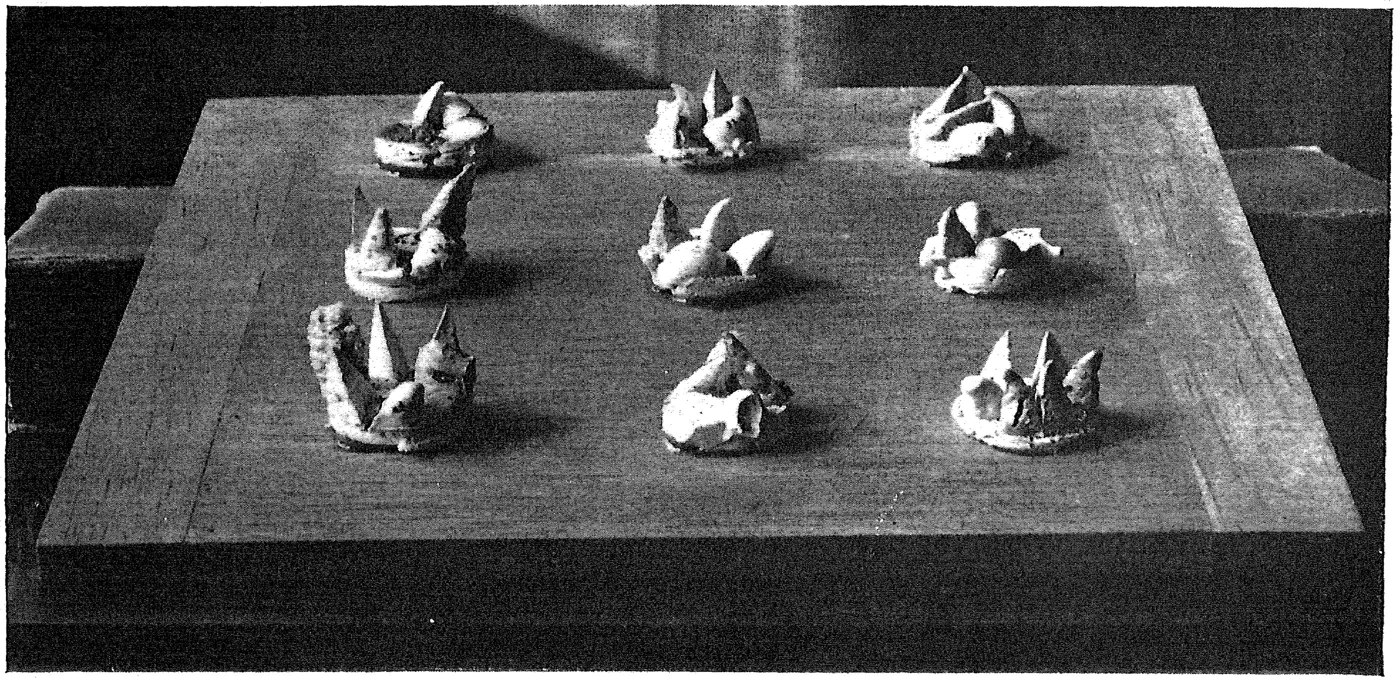
Seger Cones.*					Bischof Standard Clays.†	
No.	Composition.				Corresponding No.	Locality.
	K_2O.	CaO.	Al_2O_3.	SiO_2.		
26	0.3	0.7	7.2	72.	VII.	Niederpleis, Nassau.
27	0.3	0.7	20.	200.	......	
28	....	....	1.	10.	VI.	Cassel, Hesse.
29	....	....	1.	8.	......	
30	....	....	1.	6.	V.	Grünstadt, Palatinate.
31	....	....	1.	5.	......	
32	....	....	1.	4.	IV.	Coblence, Rhen. Province.
33	....	....	1.	3.	III.	Andennes, Belgium.
34	....	....	1.	2.5	......	
35	....	....	1.	2.	II.	Zettlitz, Bohemia.
36	Rakonitz clay-slate.				I.	Saarau, Silesia.

There are twelve standard mixtures, while Bischof only had seven standard clays. They are formed into small three-sided pyramids, $\frac{3}{8}$ inch at the base and $\frac{25}{32}$ inch high, having their respective numbers, Nos. 26 to 36, impressed on them. Their behavior in the fire is shown in Fig. 4, representing tests with clays, fire-bricks, and fire-sands, made at Berlin and sent to the writer by Mr. Cramer, to serve as a guide at the beginning of the present work. Fig. 5, with legend, describes their behavior in detail, the different changes during fusion being indicated by the terms "beaded" (the apex only showing signs of fusion), "shortened" (the pyramid bending or sinking and

* Seger, *Thonindustrie Zeitung*, 1888, p. 163. Seger-Cramer, *Thonindustrie Zeitung*, 1893, p. 1252.

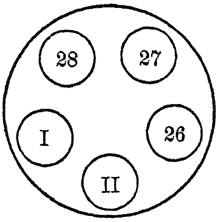
† For analyses see *Trans.*, xxiv., p. 45.

Fig. 4.



Tests of Fire-clays, Fire-bricks and Fire-sand.

Fig. 5.

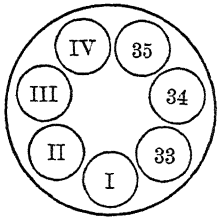


SEGER CONES.

26 GLOBULAR.
27 SHORTENED.
28 WELL-BEADED.

CLAY SAMPLES.

I. COMPLETELY FUSED AND ABSORBED BY BOTTOM LINING, MUCH BELOW CONE 26.
II. EQUAL TO CONE 27.

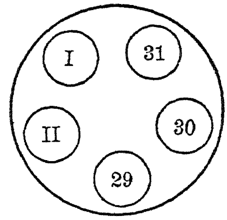


SEGER CONES.

33 GLOBULAR.
34 SHORTENED.
35 UNCHANGED.

FIRE-SAND SAMPLES

I. EQUAL TO CONE 33.
II., III. AND IV. STAND BETWEEN CONES 34 AND 35.

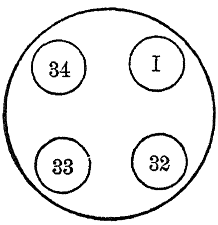


SEGER CONES.

29 GLOBULAR.
30 SHORTENED.
31 SLIGHTLY BEADED.

FIRE-CLAY SAMPLES

I. EQUAL TO CONE 30.
II. EQUAL TO CONE 30.

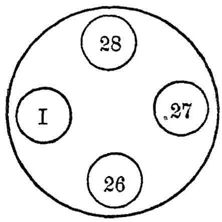


SEGER CONES.

32 GLOBULAR.
33 SHORTENED.
34 SLIGHTLY BEADED.

FIRE-BRICK SAMPLE

I. EQUAL TO CONE 33.

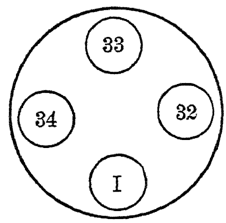


SEGER CONES.

26 GLOBULAR.
27 SHORTENED.
28 WELL-BEADED.

FIRE-CLAY SAMPLE

I. EQUAL TO CONE 28.

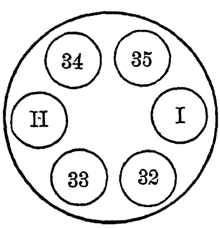


SEGER CONES.

32 LENTICULAR.
33 GLOBULAR.
34 SLIGHTLY BEADED.

FIRE-CLAY SAMPLE

I. EQUAL TO CONE 33.

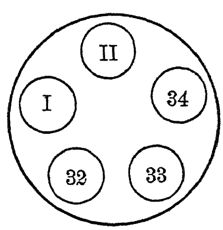


SEGER CONES.

32 SHORTENED.
33 SLIGHTLY BEADED.
34 FRITTED.
35 UNCHANGED.

FIRE-BRICK SAMPLES

I. EQUAL TO CONE 32.
II. STANDS BETWEEN CONES 32 AND 33.

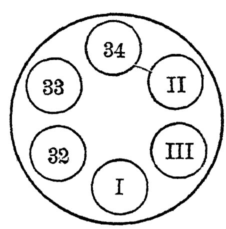


SEGER CONES.

32 LENTICULAR.
33 SHORTENED.
34 BEADED.

FIRE-SAND SAMPLES

I. STANDS BETWEEN CONES 33 AND 34
II. EQUAL TO CONE 34 OR STANDS SLIGHTLY LOWER.



SEGER CONES.

32 SHORTENED.
33 SLIGHTLY BEADED,
34 UNCHANGED.

FIRE-CLAY SAMPLES

I. AND II. STAND BETWEEN CONES 32 AND 33.
III. EQUAL TO CONE 33

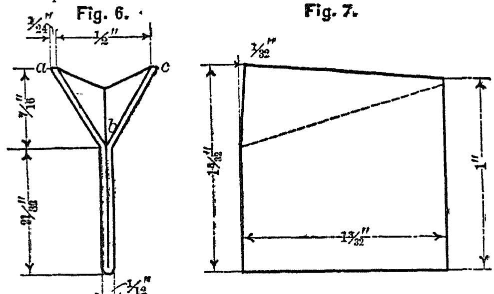
DIAGRAMS OF TESTS, SHOWING BEHAVIOR OF CONES AND SAMPLES.

bulging toward the base), "globular" (the pyramid becoming round or oval), and "lenticular" (more or less complete fusion, the sample being sometimes slightly raised in the middle, sometimes slightly depressed, sometimes completely absorbed by the porous bottom-lining of the crucible).

The writer cannot agree with Bischof's* adverse criticism of these cones. After having melted down close upon 200 Seger cones in 70 separate experiments, he has still to find one instance where the cones did not fuse in the order indicated by their numbers.

The mould and scraper used in making pyramids from the sample to be tested are shown in Figs. 6, 7, 8, and 9. The

Fig. 6. Fig. 7.



CONE-MOULD

mould, Figs. 6 and 7, consists of a small oblong piece of sheet-brass, folded crosswise, with two of the corners turned outward just far enough to form an inclined trough and soldered in the middle, the solder being filed smooth. The scraper, Figs. 8 and 9, is a piece of sheet-zinc pointed at one end to fit accurately into the triangle, *a b c*.

The dimensions here shown should be slightly reduced if the test-cone is to correspond exactly to the size of the Seger cone. The mould, Figs. 6 and 7, produces a three-sided pyramid, 0.5 inch at the base and 1 inch high, while the corresponding measurements of the Seger cone are $\frac{3}{8}$ inch and $\frac{25}{32}$ inch. The difference of size seems to make no perceptible difference in result; but not more than three of the larger cones can be put into the crucible at once—which is a decided disadvantage.

* *Thonindustrie Zeitung*, 1893, p. 1337; 1894, p. 89. See also Aaron's rejoinder, *Thonindustrie Zeitung*, 1893, p. 1337; 1894, p. 90.

4. *Method of Working.*—In preparing refractory material for a test, it is slightly calcined (if it contains any organic matter), and, when cool, moistened on a glass plate with water containing 5 per cent of dextrine and worked with a spatula to a stiff paste; plastic fine-grained clay requiring less water than sandy and open-grained. The paste is now pressed with a spatula into the mould, which is placed over the scraper. This is slowly pushed upward and thus releases the sample pyramid. It is well to slightly oil the upper part of the mould to prevent the clay from adhering and the top of the pyramid from bending over. The sample is now air-dried over night, and then slowly heated on an iron plate to drive off the rest of the

Fig. 8.

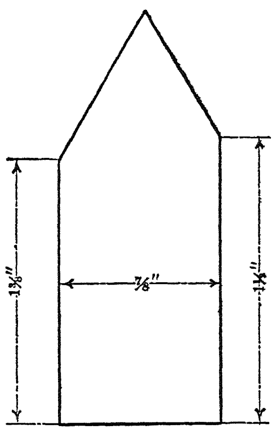
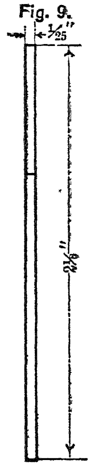


Fig. 9.



SCRAPER

moisture. Usually two dried samples are placed with three Seger cones in a crucible. To hold the cones in position refractory clay is first introduced to a depth of say ⅜ inch and then pressed down with a round stick, say 1⅛ inches in diameter and 4 inches long. With samples melting below Seger cone No. 34, the bed consists of clay melting at a temperature indicated by Seger cone No. 35; with samples more refractory than Seger cone No. 34, the bed consists of equal parts of kaolin and alumina. In making the clay bed, one-half is pressed down firmly before the rest is added. This is then tamped in gently, leaving it softer than the bottom, so that when the cones are introduced the latter will be undisturbed

and the whole firm. In putting the pyramids in position a pair of straight steel pincers was used, one arm of which had been flattened out and then bent into a trough shape.

The fuel best suited for the furnace is gas-carbon, as it is hard, dense and contains less than 0.1 per cent. of ash. It was crushed in a Gates' laboratory-crusher to pass a 2-mesh sieve, and then screened on a 4-mesh sieve to remove the fines. In order to ignite the carbon about 200 grammes of charcoal are required. This was put through the same crusher, everything passing through a 2-mesh sieve being rejected. The fire is started with about 30 grammes of newspaper.

A pair of cupel-tongs with points bent at right angles to the arms and slightly curved to fit the cylindrical form of support, crucible and lid, were found convenient. When the crucible is in position the blast is slowly started and the paper ignited and pressed down into the furnace with the tongs, to be followed by charcoal and gas-carbon. The pressure of the blast is gradually increased, and when nearly all the carbon has been consumed and the crucible becomes visible (a blue glass being necessary to permit the watching) the blast is shut off and the crucible removed, the support usually adhering to it. The different temperatures are obtained by varying the the amount of gas-carbon and the blast. In the Berlin laboratory, foot-bellows, 20 inches in diameter, are used; at first they are trodden 25 times a minute, the number of treadings being afterwards increased to 50, and a test lasting about one hour. The use of foot-bellows being rather awkward and time-consuming, a small Sturtevant fan furnished the blast in the writer's experiments, the connection between the blower and the rubber hose attached to the blast-pipe of the furnace being made with a tuyere-bag. Into the rubber hose was inserted a glass tube, and this connected by rubber tubing with a U-shape water-gauge; the pressure of the blast was regulated by tightening or loosening a clamp enclosing the tuyere-bag.

The first step in experimenting with the furnace was to find out the amount of gas-carbon and the pressure required for the different Seger cones. Table III. gives the leading results obtained. It took about six minutes to feed in the charge, fresh charcoal or gas-carbon being given when the glow had penetrated the preceding charge of fuel. The pressure of the blast

TABLE III.—*Furnace-Tests of Seger Cones.*

Seger Cone.			Fuel-Charge.				Blast-Pressure.								Furnace.			
							Inches water at minutes after starting.								Initial Condition.	Appearance of, in minutes after starting.		
No.	Form after Fusion.	Luster after Fusion.	Paper, grms.	Charcoal, grms.	Gas-carbon, grms.	Time of feeding, Minutes.	Inch.	Min.	Inch.	Min.	Inch.	Min.	Inch.	Min.		First CO-flame.	Steady CO-flame.	Crucible.
26	Beaded	Dull	30	200	900	6	0.5	1	1	3	...	...	...	...	Cold	8	10	37
26	Globular	Dull, vitreous	30	200	925	7	0.5	1	1	7	...	...	...	...	Cold	13	18	35
26	Lenticular	Vitreous	30	200	950	7	0.5	1	1	7	...	...	...	...	Cold	13	18	38
27	Globular	Dull	30	200	925	7	0.5	1	1	7	...	...	...	...	Cold	13	18	35
27	Globular	Dull, vitreous	30	200	950	7	0.5	1	1	7	...	...	...	...	Cold	13	18	38
27	Lenticular	Vitreous	30	200	995	7.5	0.5	1	1	6	1.5	17	...	...	Cold	19	23	41
28	Shortened	Dull	30	200	925	7	0.5	1	1	7	...	...	...	...	Cold	13	18	35
28	Lenticular	Dull	30	200	950	7	0.5	1	1	7	...	...	...	...	Hot	13	18	38
28	Lenticular	Vitreous	30	200	925	6	0.5	3	1	6	1.5	12	...	...	Hot	12	13	35
29	Globular	Dull	30	200	900	6	0.5	1	1	3	1.5	6	...	...	Hot	11	12	29
29	Lenticular	Vitreous, dull	30	200	925	6	0.5	1	1	3	1.5	6	...	...	Hot	12	13	35
29	Lenticular	Vitreous	30	200	1000	6	0.5	1	1	3	1.5	5	...	...	Hot	15	17	45
30	Shortened	Dull	30	200	900	6	0.5	1	1	3	1.5	6	...	...	Hot	11	12	29
30	Lenticular	Dull, vitreous	30	200	925	6	0.5	1	1	3	1.5	6	...	...	Cold	11	14	28
31	Globular	Dull	30	200	925	6	0.5	1	1	3	1.5	6	...	...	Hot	12	13	31
31	Lenticular	Vitreous	30	200	925	6	0.5	1	1	3	1.5	6	...	...	Cold	10	11	29
32	Shortened	Dull	30	200	925	6	0.5	1	1	3	1.5	6	...	...	Cold	11	14	28
32	Globular	Dull	30	200	1025	6	0.5	1	1	3	1.5	6	...	...	Cold	11	13	31
32	Lenticular	Dull	30	200	950	6	...	...	1	1	1.5	6	2	17	Hot	10	12	30
33	Shortened	Dull	30	200	1025	6	...	...	1	1	1.5	6	...	...	Hot	11	15	36
33	Lenticular	Dull	30	200	1050	6	0.5	1	1	3	1.5	6	...	...	Hot	12	13	35
33	Globular	Duil	30	200	925	6	0.5	1	1	3	1.5	6	2	17	Cold	10	14	24
34	Beaded	Dull	30	200	925	6	0.5	1	1	3	1.5	6	...	...	Hot	10	11	25
34	Globular	Dull	30	200	925	6	...	...	1	1	1.5	6	2	17	Hot	9	11	29
34	Lenticular	Dull	30	200	975	6	...	...	1	1	1.5	6	2	17	Hot	10	11	?
35	Beaded	Dull	30	200	925	6	...	...	1	1	1.5	6	2	17	Cold	?	17	26
35	Globular	Dull	30	200	925	6	...	...	1	1	1.5	6	2	17	Hot	9	11	29
35	Lenticular	Dull	30	200	1000	6	0.5	1	1	3	1.5	6	2	17	Hot	8	10	25

TABLE IV.—*Analyses and Tests of Standard American Fire-Clays.*

No.	Fire-Clay From.	Sampled by.	Analyzed by.	Al_2O_3, p.c.	SiO_2, combined, p.c.	H_2O, combined, p.c.	Total clay material, p.c.	SiO_2, non-combined, p.c.	TiO_2, p.c.	Total sandy material, p.c.	MgO, p.c.	CaO, p.c.	K_2O, p.c.	Na_2O, p.c.	Li_2O, p.c.	Fe_2O_3, p.c.	Total fluxing material, p.c.	H_2O, hygroscopic, p.c.	Organic matter, p.c.	Loss on ignition, p.c.	Total, per cent.	Corresponding Seger cone, No.
1	Athens, Texas	Ladd	Ladd	20.71	31.82	7.17	59.70	37.06		37.06	0.39	0.22	0.69	0.39	......	1.01	2.70	1.82	......	8.99	99.46	29
2	Blanford, Mass. (kaolin)	Twichell.	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	29
3	Mt. Savage, Md. (soft)	Ladd	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	29–30
4	Woodbridge, N. J. (Dixon)	Ladd	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	30–31
5	Parall, Ohio (No. 4)	Ladd	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	30–31
6	St. Louis, Mo. (Christie, raw clay)	Ladd	Ladd	21.16	26.03	8.94	56.13	38.32		38.32	0.30	0.61	0.86	0.88	......	2.72	5.37	2.62	......	11.57	99.82	31–30
7	Carbondale, Col. (kaolin)	Ladd	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	31
8	Pueblo, Col. (20 miles S. W. of)	Gilbert	Steiger	24.72	28.76*	8.63*	62.11	34.76*	0.68	35.44	0.13	0.30	trace	trace	P_2O_5 trace	0.43	0.86	1.36	0.40	10.39	100.17	31
9	Golden, Col	Ladd	Ladd	31.72	45.99	13.30	91.01	5.22		5.22	0.23	0.36	0.48	0.45	......	0.75	2.27	0.83	......	14.13	98.50	32–31
10	Parall, Ohio (No. 5)	Ladd	Ladd	28.14	32.03	10.22	70.39	26.81		26.81	0.28	0.49	0.15	0.46	......	1.67	3.05	1.23	......	11.45	100.25	32
11	Woodbridge, N. J. (quartz-kaolin)	Smock	?	17.46	16.50	6.30	40.26	57.10	0.90	58.00	......	......	0.12	0.21	......	0.54	0.87	0.50	......	6.80	99.63	32–33
12	St. Louis, Mo. (Evans and Howard)	Ladd	Ladd	23.26	27.56	10.20	61.02	31.82		31.82	0.42	0.65	0.54	0.79	......	3.24	5.64	2.74	......	12.94	98.48	32–33
13	Mineral Point, Ohio	Orton, Sr.	Lord	31.84	35.39	11.68	78.91	17.13	1.68	18.81	0.19	0.50	0.59	......	......	0.67	1.95	0.69	......	12.37	100.36	33
14	Carbondale, Col	Ladd	Ladd	31.75	36.03	12.76	80.54	17.00		17.00	0.04	0.87	0.16	0.45	......	1.06	2.58	2.42	......	15.18	100.12	33–34
15	Sciotoville, Ohio	Ladd	Ladd	40.04	43.28	13.11	96.43	0.31		0.31	0.17	0.15	0.30	0.38	......	2.40	3.40	1.40	......	14.51	100.14	34
16	Sand Hills, N. J. (Woodbridge)	Smock	?	35.94	36.20	12.10	84.24	12.20	1.50	13.70	......	......	0.08	......	......	0.96	1.04	1.10	......	13.20	100.08	34
17	Woodland, Pa	Ladd	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	34–35
18	Mt. Savage, Md. (hard)	Ladd	Ladd	38.14	41.93	12.62	92.69	5.90		5.90	0.17	trace	0.49	0.42	......	1.31	2.39	1.18	......	13.80	100.98	34–35
19	High Hill, Md	Ladd	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	34–35
20	Woodbridge, N. J	Ladd	Ladd	36.85	43.46	14.03	94.34	1.92		1.92	0.83	0.15	0.21	0.28	......	0.99	2.46	1.34	......	15.37	98.72	35–34
21	Sayreville, N. J	Smock	?	38.66	41.10	13.55	93.31	3.10	1.20	4.30	......	......	0.28	0.18	......	0.74	1.20	1.00	......	14.55	99.81	35
22	Portsmouth, Ohio	Orton, Jr.	Orton, Jr.	39.93	45.05	14.15	99.13	with clay	......	......	0.18	0.07	0.21	trace	......	0.47	0.93	......	......	......	100.06	35
23	Sciotoville, Ohio	Orton, Sr.	Mariner	34.78	52.00	10.96	97.74	with clay	......	......			0.78			1.48	2.26	......	......	......	100.00	35
24	Burt's Creek, N. J. (South Amboy)	Smock	?	38.34	42.90	13.50	94.74	1.50	1.20	2.70	......	......	0.26	0.18	......	0.86	1.30	1.10	......	14.60	99.84	35
25	Woodbridge, N. J. (Spa Spring)	Smock	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	......	36–35

* Calculated. Analysis of clay No. 8 was made in the laboratory of the United States Geological Survey. Analyses of clays Nos. 11, 16, 21 and 24 are copied from the *Report on Clays* of the Geological Survey (Trenton, 1878, Nos. 22, 14, 20 and 19, p. 298), as the clays there represented, according to Prof. J. C. Smock, are of the same composition as those tested. Analyses Nos. 13 and 23 are taken at the suggestion of Prof. Edward Orton, Sr., from vol. v. of the Geological Survey of Ohio (Columbus, 1884, p. 648), Nos. 1 and 6, in the table there given. In Nos. 8, 11, 13, 16, 21 and 24, the hygroscopic water is included in the total.

was slowly increased. Starting with just enough air to permit of introducing the ignited paper without its blazing up too violently, it was raised to $\frac{1}{2}$ inch when the first scoopful of charcoal had been given, to 1 inch when all the charcoal and some gas-carbon had been fed, and then gradually to the required pressure, the highest being 2 inches. When the pressure required for a certain experiment had been reached, it was maintained at that point until the end by increasing the amount of air in proportion to the burning away of the carbon. The result of this is that toward the end of an experiment the gas-carbon is burnt quickly and thereby the temperature greatly raised. This is important if the samples are to undergo the desired change. The table shows the different effects obtained with different weights of gas-carbon and varying pressures of blast. Of course, it must be borne in mind that the figures depend upon the character of the gas-carbon and the condition of the furnace, and should therefore vary when these differ from those used. It has been previously said that, after making some 60 experiments, the lining of the furnace showed signs of becoming vitrified, which slightly enlarged the diameter of the hottest zone, with the result that more fuel or a higher pressure of blast, or both, were required. In making this kind of experiment, the first thing is to standardize the furnace, if the expression be allowable, by following in a general way the data given in the table and then keeping a tabular record of all the experiments made afterward, the latest successful experiments serving as guides for future work. In the table the condition of the furnace when the experiment was made is given as cold or hot; the former means that no work had been done with the furnace on the day of the experiment, the latter that one or two experiments had preceded the one in question.

5. *Results with American Fire-Clays.*—In Table IV. are recorded the results obtained in testing standard American fire-clays by Seger's method. Of the 25 samples, fifteen were received from Dr. G. E. Ladd, a geologist who, having made the study of clays a specialty, devoted much time and money in visiting the principal deposits and collecting samples (weighing each about 50 pounds) for investigations of his own. These samples are specially good, being taken in a systematic way by

a disinterested expert. The results of the fire-tests show, therefore, the true character of the beds in 1893. How this may differ from samples taken at an earlier date is seen by comparing tests Nos. 15 and 23, both representing Sciotoville clay; No. 23 the clay of 1884 and No. 15 that of 1893. Only 8 of Dr. Ladd's samples have so far been analyzed; the analyses of the other 7 will appear later. Prof. J. C. Smock kindly furnished the writer with abundant material from the representative clay-beds of New Jersey. The analyses quoted are not actually those of the clays that were tested, but are sufficiently close to permit their being used. Of the three Ohio clays received from Professors Edward Orton, Sr., and Edward Orton, Jr., only the sample from Portsmouth is represented by the actual analysis. The clay from Pueblo, Col., received from Mr. G. K. Gilbert, of the United States Geological Survey, is represented by the true analysis. The last column of Table IV. shows the comparative fire-resisting values of the clays. They range from Seger cone No. 29 to No. 35 and above; a pyramid of Spa Spring washed clay from Woodbridge, N. J., melting down only to the size of a pea, when Seger cone No. 35 had been completely melted.

In addition to these tests of standard clays, five samples of low-grade fire-clays from Ohio and Pennsylvania, kindly furnished by Prof. Edward Orton, Jr., have been similarly tested. Table V. gives the analysis and results:

TABLE V.—*Analyses and Tests of Low-Grade Fire-Clays.*

Number.	Fire-clay Sampled and Analyzed by Prof. Edward Orton, Jr. from	Al_2O_3. Per cent.	SiO_2, Total. Per cent.	H_2O, combined. Per cent.	Total Clay Material. Per cent.	MgO. Per cent.	CaO. Per cent.	K_2O. Per cent.	Na_2O. Per cent.	Fe_2O_3. Per cent.	Total Fluxing Material. Per cent.	H_2O, hygroscopic. Per cent.	Loss on ignition. Per cent.	Total, including hygr. H_2O. Per cent.	Corresponding Seger cone. No.
1	Zanesville, O........	22.95	64.26	6.74	93.95	0.37	0.45	1.81	0.15	1.28	4.06	2.05	8.79	100.06	26
2	Salineville, O.......	26.60	56.44	7.57	90.61	0.63	0.47	3.20	0.26	2.00	6.56	2.48	10.05	99.65	27–26
3	New Brighton, Pa.	24.88	63.55	6.96	95.39	0.47	0.56	2.27	......	1.17	4.47	1.38	8.34	101.24	28
4	Zanesville, O........	21.13	66.21	6.29	93.63	0.18	0.51	1.42	0.38	1.28	3.77	1.65	7.94	99.05	29–28
5	East Palestine, O..	25.12	59.54	7.75	92.41	0.51	0.57	1.95	......	1.57	4.60	2.63	10.83	99.64	29–28

It will be seen that the clays of Table V. correspond approximately in refractory quality to Seger cones Nos. 26, 27, and 28 (although No. 27 is not accurately represented), thus completing, with Table IV., the range covered by the Bischof standards, as shown in Table II.

The Nomenclature of Zinc-Ores.

BY WALTER RENTON INGALLS, NEW YORK CITY.

(Florida Meeting, March, 1895.)

THE ores of zinc which are important as sources of that metal are of two classes, viz., the sulphide and the oxidized. The latter includes six varieties: zincite (the red oxide) and franklinite (the oxide of zinc, iron, and manganese), which are found abundantly only in New Jersey; and the hydrous and anhydrous carbonates, the hydrous and anhydrous silicates, which are of widespread occurrence.

The hydrous carbonate is known mineralogically as hydrozincite, zinconise, or zinc-bloom; the anhydrous silicate is recognized as willemite. With respect to the anhydrous carbonate and the hydrous silicate, however, there is a confusion of name which is of old standing. Attempts to clear it away were long ago made by the mineralogists with the result that there is now a more or less national uniformity of nomenclature; but there is still an international disagreement, sometimes very perplexing and always leading to inexactness in expression, which may be considered an adequate excuse for this return to a time-worn subject.

The name *calamine*, together with *Galmei* of the Germans, is commonly supposed to be derived from καδμεια, which was used by the Greeks to designate the peculiar kind of ore employed with copper in their brass-making, and also the accretions which formed in the brass-founder's furnaces. Agricola, however, says that it is from *calamus*, a reed, in allusion to the appearance of the material, *cadmia fornacum*, which collected on the furnace-walls. But whatever the derivation of the word, it was used until within one hundred years to include all the oxidized

ores and compounds of zinc, both natural and artificial. Indeed, the difference between the carbonates and the silicates does not seem to have been suspected before 1780, when Bergmann published an account of certain experiments upon them; and it was not until 1803 that their true composition was made known by Smithson, and all doubts as to their being distinct mineral species were cleared away.

The naming of these minerals is described by Dana in his *System of Mineralogy* (1892), pp. 548–549. In 1807, Brongniart called the silicate *calamine*, leaving for the other mineral its chemical name, *zinc carbonaté*, by which it continued to be known until, in 1832, Beudant dubbed it *smithsonite*. In 1852, Brooke and Miller, with no good reason, reversed these names, and thus led to the confusion which still exists. On account of this confusion, Kenngott, in 1853, introduced for the silicate the name *hemimorphite*, concerning which Dana severely says that such "innovations should have no favor."

At the present time American usage follows Dana, calling the anhydrous carbonate smithsonite and the hydrous silicate calamine. English mineralogists, on the contrary, generally employ calamine to designate the anhydrous carbonate, referring to the hydrous silicate as *electric calamine*. The application of the name smithsonite to the hydrous silicate by Brooke and Miller had a certain following in their time (*vide* Greg and Lettsom, *Mineralogy of Great Britain*, 1858), but no longer obtains. On the Continent of Europe, however, the equivalent names, *calamine* and *galmei*, are used in common parlance, especially in the zinc industry, to include the four mineral varieties, carbonates and silicates, hydrous and anhydrous. In Germany, many mineralogists use the nomenclature adopted by Dana (*vide* Von Kobell, *Geschichte der Mineralogie*, and Naumann, *Elemente der Mineralogie*), but most writers on technical subjects employ *galmei* as a class-name only, designating the silicates as *Kieselgalmei*, and the anhydrous carbonate as *edler Galmei*, *Smithsonite*, or *Zinkspath* (zincspar). French writers avoid confusion by using the chemical terms *zinc carbonaté* and *zinc silicaté*, although in France, as in Germany, *calamine* (*galmei*) is employed by mineralogists as a purely scientific name for a distinct species—the hydrous silicate. The general meaning that the word calamine has on the Continent is probably a survival

of the custom of the time when no difference in the oxidized ores of zinc was recognized.

Without regard to the rightful claim to these names according to the rules of nomenclature that have been laid down by eminent mineralogists, it seems that when a mineralogical name is popularly adopted to express a meaning different from that ascribed to it in science, and from such popular usage it passes into the dictionaries of the language, its more restricted use in science should be discontinued. Otherwise, misapprehension is likely to result, to avoid which such clumsy explanatory clauses as "calamine, meaning anhydrous carbonate of zinc," or "asbestos, meaning fibrous hornblende," must be introduced. Therefore such names should be left exclusively as the industrial or class-names which they have become, and new names should be adopted for the species.

In the case of zinc-ores there is no confusion as to the meaning of hydrozincite or of willemite. It may be affirmed also that the significance of smithsonite is clearly understood, since it is now never used for anything but the anhydrous carbonate of zinc. The hydrous silicate, however, should have a new name. Perhaps it would be simplest to call it *hydrowillemite*, which would be explicit, and would convey its meaning at first sight.

The White Phosphates of Tennessee.*

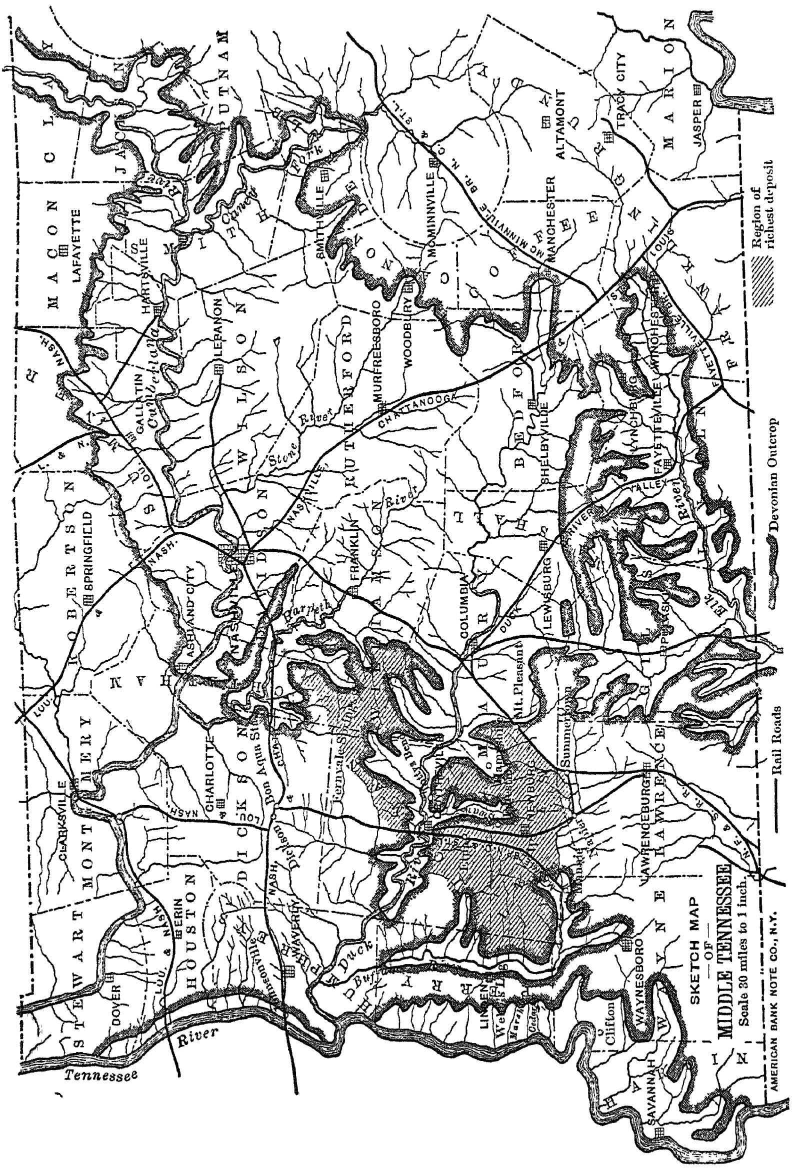
BY CHARLES WILLARD HAYES, WASHINGTON, D. C.

(Florida Meeting, March, 1895.)

SHORTLY after the discovery of black phosphate on Swan creek, in Hickman county, Tennessee, prospectors familiar with the Florida phosphate came to the region and began the search for rock similar to that found in Florida. Among these was Mr. E. Slattery, who located at Linden, in Perry county. He gave a piece of the Florida rock to Mr. C. C. Sutton, and the latter discovered on Tom's creek a deposit which bore a strong

* Published by permission of the Director of the United States Geological Survey.

resemblance to the sample. This proved to be the phosphate breccia, described below, and the country was carefully examined to determine the extent of the deposit. A short time



after, Mr. P. L. Smothers discovered in the same way, on Red Bank creek, another deposit bearing even stronger resemblance

to the Florida rock. This was the Carboniferous bedded phosphate.

As indicated above, there are two varieties of the white phosphate, distinct in appearance, mode of occurrence and origin, namely (1), the brecciated rock and (2) the white-bedded rock. Both varieties, so far as known, are restricted to Perry county, although future prospecting may greatly extend their range. For the localities mentioned in this paper, the reader is referred to the accompanying map, originally published with the paper of Messrs. Meadows and Brown, on the "Phosphates of Tennessee" (*Trans.*, xxiv., 582), read at the Bridgeport meeting, October, 1894. This map, though covering territory not referred to in the present paper, is reprinted to serve the convenience of the reader, while avoiding the expense of a new engraving.

I.—White Breccia Phosphate.

Location of the Deposits.—This variety is most highly developed on Tom's creek and upon the west side of Buffalo river, north of Linden. It has been reported, also, from Spring and Lick creeks, south of Tom's, and from Rones's creek, on the north. These streams are northwest of Linden, in Perry county, and flow westward to the Tennessee river in rather wide valleys, from which the intervening remnants of the plateau rise with gentle, rounded slopes. The phosphate-breccia is found upon these slopes within a vertical range of from 30 to 50 feet, and following the windings of the valley-sides with slight variation in altitude. Its upper limit appears to be the outcrop of the Devonian black shale; but this is somewhat difficult to determine, since the overlying chert deeply covers the surface, and outcrops of the black shale are extremely rare. The outcrops of the phosphate-rock are not continuous. In some places occasional boulders only are found, while at others the material covers the entire surface. No work has been done to determine its depth. At some places it appears to rest upon an eroded surface of Silurian limestone, but more often it is simply imbedded in the residual chert. The rock nowhere shows any trace of bedding, either within its own mass or in its relations as a whole to the formations with which it comes in contact.

Composition and Physical Appearance.—The phosphate-rock occurs in large irregular masses, composed of small angular fragments of Carboniferous chert imbedded in a matrix of phosphate of lime. The chert fragments vary in diameter from a fraction of an inch to three or four inches. They are in every respect similar to the fragmental chert which so abundantly covers the hillsides. The phosphatic matrix, when unstained by exposure to the weather, is generally white or slightly reddish and rather soft—somewhat harder than compact chalk. In some cases the phosphate shows a laminated structure, as though the cavities between the chert fragments had been filled by the deposition of successive layers of material from solution, and sometimes the cavities were only partially filled. Where this concentric structure is shown the phosphate is more dense and also purer than the structureless variety.

Not enough prospecting has yet been done to afford a basis for an accurate estimate as to the available amount of phosphate of this variety, but some idea may be gained by considering the length of actual outcrop. Assuming for the deposit a uniform width of 75 feet, it is estimated that in the territory thus far known there are between 40 and 50 acres which would be actually productive. On a very moderate assumption as to the depth of the deposit, this would yield several hundred thousand tons of the material. Of course, careful prospecting may show this estimate to be far above or below the mark.

It is quite probable that further exploration will extend the known area within which this breccia occurs. On long exposure to the weather the matrix crumbles away, freeing the chert fragments, which cover the surface and are indistinguishable from other portions of the widespread and almost universal mantle of chert covering this portion of Tennessee. Hence, where only a few boulders of the breccia are now found at the surface, there may be a more or less continuous deposit beneath the superficial mantle. It is also probable that other deposits may exist which now present at the surface no indication whatever of their presence. This would be especially likely of such as contain an exceptionally small proportion of chert. In most of the breccia examined, the chert fragments

make up about 50 per cent. of the rock. At the Ledbetter place on Tom's creek and also near Beardstown, some rather large masses were observed which appeared to be nearly free from chert. Closely resembling some forms of travertine which are being deposited by calcareous springs, they suggest an analogous origin. These detached masses may be portions of a more extensive deposit of similar material, almost entirely covered by the mantle of chert, as suggested above.

Utilization of the Deposit.—Analyses of the breccia-matrix show it to be a high-grade phosphate, and it would probably be found, in most cases, when carefully separated from the associated chert, to contain 80 per cent. of lime phosphate. A sample of the travertinoid rock, from the vicinity of Beardstown, gave 80.92 per cent. of lime phosphate. Neglecting exceptional occurrences, the ordinary rock-matrix and chert together give about 40 per cent. of lime phosphate. This is too low a percentage to be utilized by present methods in the manufacture of fertilizers. The problem for the mining engineer is therefore to devise some cheap method by which the crude material can be freed from a portion, at least, of the chert. As stated above, the matrix is rather soft, and shows a tendency to crumble to a powder when the rock is crushed. The chert, on the other hand, is much harder, and breaks into smaller fragments without shattering or crumbling. Hence it appears to the writer probable that if the rock were crushed so as to pass through an inch-mesh, and then passed over a screen with a quarter- or third-inch mesh, the greater part of the phosphate would pass through with comparatively little chert. If only half the chert were thus removed, the proportion of lime phosphate would be raised from 40 to 53 per cent. This would bring it above the limit of availability for use by methods now employed in the manufacture of fertilizers.

Origin of the Deposits.—From the appearance of these deposits and their relations to adjacent formations, there can be little doubt that they are recent and superficial, the result of the leaching of the black phosphate and redeposition near its outcrop. It seems probable that surface-water, containing a large proportion of carbonic acid, reached the black shale through the overlying porous covering of chert. The lime phosphate associated with the black shale was dissolved by the percolating acidulated water, which subsequently reached the

surface at the outcrop of the shale. The dissolved phosphate was then redeposited, partly in the interstices of the fragmental chert covering the surface, and partly as a solid deposit with but slight intermixture of chert. The former mode of deposition produced the more abundant breccia, while the latter gave rise to the travertinoid masses. Subsequent erosion has doubtless lowered the valleys throughout the region, and removed much of the phosphatic deposits thus formed.

If this theory of its formation be correct, the deposits will be found only near the surface in shallow pockets; and although there is unquestionably a large amount of the material in sight, mining will be attended with the uncertainties which invariably accompany the working of pocket-deposits.

II.—White Bedded Phosphate.

Location of the Deposit.—So far as at present known, this variety is confined to a small area in Perry county. It has been found on Red Bank and Terrapin creeks, which flow eastward into Buffalo river. The extreme outcrops lie within an area about three miles long by a little over a mile broad. Within this area numerous outcrops occur, though the rock has not been traced continuously from one to another, and it is only inferred that they form a continuous bed.

The northward dip of the strata carries the Devonian down to the level of Buffalo river, above the mouth of Red Bank creek, so that the intersected plateau is here composed wholly of the lower portion of the Carboniferous, which consists mostly of chert in calcareous, sandy shale.

The phosphate is found about 70 feet above the Devonian black shale, interbedded with the Carboniferous chert. At the Spencer place, on Red Bank creek, the phosphate and chert crop out in a ledge about 20 feet high, above which occur numerous though not continuous exposures, making a total thickness of at least 30 feet. Of this, the lower 20 feet consists of alternate beds of phosphate and chert. The latter appears in lenticular beds, from 4 to 12 inches thick, its contact with the phosphate being somewhat indistinct. Portions of the phosphate are highly siliceous, approaching chert in appearance and probably in composition. The appearance is that of an incomplete replacement of the chert by the phosphate. According to an approximate estimate of this portion of the formation, the

chert beds appear to make up about 30 per cent. of the mass, the remainder being more or less siliceous phosphate.

The upper 10 feet, although not so well exposed, appears to be made up almost entirely of obscurely bedded phosphate, without any considerable portion of chert. The phosphate in this part of the formation is also whiter, softer, and evidently less siliceous.

In Stone-Quarry Hollow, on the south side of Terrapin creek, the phosphate is exposed about 40 feet in thickness. As upon Red Bank creek, the lower portion consists of alternating beds of stony chert and hard siliceous phosphate, while the upper portion, perhaps 10 or 15 feet in thickness, is free from chert beds, or, if present, they do not appear at the surface.

At the Myatt place, on Terrapin creek, the formation is at least 30 feet in thickness; but the bedding is less distinct than at the points above described, and there does not appear to be so marked a difference between the upper and the lower portion, although this may be due to less complete exposure.

Physical Appearance.—The phosphate-rock is much harder than ordinary lime phosphate, and breaks with an extremely rough, irregular surface. It has a finely granular structure, some portions resembling a very fine quartzitic sandstone, but grading into translucent chert. The patches of gray chert surrounded by the white granular rock give a mottled appearance to the fresh surfaces. The chert is not in the form of sharply-defined fragments, such as occur in the phosphatic breccia, but merges into the granular ground-mass, which consists of a skeleton of silica, holding soft, white lime phosphate. It is the presence of this siliceous skeleton which gives the apparently granular material its great hardness. Many small, irregular cavities occur in the rock, and these are generally lined with minute quartz crystals. Thin sections of the phosphatic rock exhibit under the microscope a more or less continuous ground-mass of chalcedonic or crypto-crystalline silica. This is only slightly crystalline, having very little double refraction. Within this ground-mass are rhombohedral crystals of lime phosphate. In portions of the rock, which appear as compact chert, the crystals are very minute (often less than .007 mm. in diameter) and widely scattered; but they are perfect, sharply defined rhombohedrons. In the granular portions of the rock the crystals are larger, appearing as sections

of rhombohedrons, which are not perfectly independent, but are segregated into irregular groups, surrounded and penetrated by the ground-mass of silica. These rhombohedral crystals have the external form of calcite, but are entirely isotropic, and hence are not calcite. The smaller crystals are quite clear and transparent, while the larger are composed of an aggregate of very minute transparent grains, with fine dust-like opaque particles, probably iron oxide. Many aggregates of similar transparent grains, but without definite crystal-outlines, occur in the ground-mass. Analyses of the rock make it evident that the materials forming the crystals and the granular aggregates must be lime phosphate. The crystal-forms are evidently those of calcite, and the crystals are therefore, in all probability, pseudomorphs, in which the lime phosphate has replaced the carbonate. The presence of a small amount of carbonate, shown in the table of chemical analyses below, indicates that the replacement has not been complete.

Chemical Composition.—The following analyses * give a fair idea of the composition of this variety of phosphate.

Analyses of Tennessee White Bedded Phosphate.

	14*c*.	14*i*.	14*k* and *l*.	14*m*.	15*d*[1].	15*d*[2].
Silica, SiO_2	61.34	49.43	54.30	54.88	50.18	56.46
Lime, CaO	20.30	26.40	22.87	22.76	25.57	22.01
Phosphoric acid, P_2O_5	12.55	15.12	14.86	15.30	15.21	13.15
Corresponding to:						
Lime phosphate, $Ca_3P_2O_8$	27.40	33.00	32.45	33.40	33.20	28.60
and						
Lime carbonate, $CaCO_3$	9.75	15.21	9.36	8.23	13.45	11.56

14*c*.—Stone-Quarry Hollow, south of Terrapin creek. Phosphate and chert 2 feet from base of exposure; represents a bed 8 inches thick between thinner beds of chert.

14*i*.—Stone-Quarry Hollow. Represents the upper 10 feet of the deposit, above the interbedded chert and phosphate.

14*k* and *l*.—Stone-Quarry Hollow. Represents 10 feet of outcrop, 20 to 30 feet above its base.

14*m*.—Stone-Quarry Hollow. Represents 6 feet of outcrop, 30 to 36 feet above its base.

15*d*[1] and 15*d*[2].—Red Bank creek and Spencer place. Represents upper 10 feet of the deposit, from 20 to 30 feet above the base of the exposure.

* Analyses made for the United States Geological Survey by the chemical department of Columbian University, Washington, D. C., under the direction of Prof. C. E. Monroe.

Only the silica, lime and phosphoric acid were determined; but in each case there was an excess of lime over that required for combination with the phosphoric acid to form the neutral phosphate, and this excess was regarded as present in the rock as carbonate. Considering the lime as part carbonate and part phosphate, the proportions of these compounds, together with the silica, amount to from 96 to 98 per cent. of the rock. The remaining 2 to 4 per cent. is probably iron and alumina, which were not determined.

Utilization of the Deposit.—It will be seen from these analyses that the content of the lime phosphate is too low for utilization by methods at present employed in the manufacture of fertilizers; 50 per cent. rock being about the lowest grade now used. Whether other processes may be devised for utilizing this low-grade material is a question which cannot be answered now. The abundance of the rock, the ease of mining and the availability of cheap water-transportation to points of consumption are important factors in the problem. But, whether utilized or not, this deposit is of interest as suggesting the possibility of other deposits of higher grade in rocks not hitherto suspected of containing phosphates, namely, the widespread Carboniferous chert-formations of Tennessee, Alabama, Kentucky, Missouri and Arkansas. From this point of view the origin of the deposit becomes a matter of considerable importance; for it is scarcely credible that the conditions under which this deposit was formed should not have been present elsewhere in this extensive region.

Origin of the Deposit.—Is the phosphate an original deposition accumulated during the deposition of the accompanying chert? The characteristics which point to original deposition in the case of the black Devonian phosphate are wholly absent here. Although some portions of the Fort Payne chert are highly fossiliferous, others are entirely barren, and, unlike the black phosphates, no traces of organic remains were observed in these or the associated cherts. Moreover, the great thickness of this deposit and its apparently local development are in striking contrast with the very wide distribution of a comparatively thin bed in the case of the black phosphate. If not an original deposit, this must be a secondary impregnation, and partial replacement of some original constituent, by lime phos-

phate. The microscopic structure of the rock affords strong evidence, if not conclusive proof, of this secondary replacement of the originally-contained calcite by secondary phosphate. The source of the phosphoric acid is not so easily determined as the fact that replacement has occurred. Limestones generally contain a small amount of phosphoric acid, and some of the overlying Carboniferous limestones contain a very considerable percentage. This seems the most probable source, although there is a possibility that the phosphate may have come from older Devonian and Silurian rocks, raised to a higher level by the gentle folding which the strata of the region have suffered.

Probably the replacement was a phase of weathering, and took place after the superincumbent strata had been largely removed, so that percolating waters had access to these beds. It is impossible at present to say what the particular conditions may have been which determined the local accumulation of phosphate at this point, and no sufficiently detailed examination has been made to decide whether or not these conditions can be recognized and so definitely formulated as to be of value in future prospecting. It will be readily understood that the study which has been given thus far to these interesting deposits is wholly inadequate to answer the many questions suggested.

Geological Sketch of Florida.

BY E. T. COX, ALBION, FLORIDA.

(Florida Meeting, March, 1895.)

THE peninsula of Florida is remarkable for the uniform character and simplicity of its geological structure.

Major Henry Whiting, of the U. S. Army, was one of the first to give an account of the physical features and rocks of Florida, in an article published in 1839, in the *American Journal of Science* (vol. xxxv., p. 47). He says:

"The rocks found *in situ* are all calcareous, though siliceous boulders are occasionally seen, and nodules of hornstone are here and there mingled with the limestone, which elicit sparks and are sometimes used by the Indians for flints."

"The coast as far as Cape Florida is alluvial, a seeming mass of comminuted shells, resting on a rocky formation, composed also of shells, more or less broken and abraded. From Cape Florida the formation is mostly coralline, the keys being of that character. As high as Indian river inlet, the beach is still formed of shells mingled with some sand, while about Cape Canaveral the sand predominates."

He believes the whole beach to be calcareous. He describes the coquina quarries of St. Augustine, and the shell formations of the upper St. Johns, as made up chiefly of *Helix*.

"On Black Creek, west of the St. Johns, a porous, rotten limestone appears, and this is said to be the characteristic of the rock formations throughout the western part of the peninsula. Hence the many sink-holes which appear in these regions, and the disappearance of streams for many miles beneath the surface, while others come forth in all their fullness at once."

He also mentions the large limestone springs, impregnated with sulphuretted hydrogen.

In an article published in 1846, in the same journal, John Allen, Lieutenant of Artillery in the U. S. Army, describes a hard, white limestone, with an earthy texture, as occurring at Fort Brooke, at the head of Tampa Bay, saying that, "in places it is soft and friable, much resembling chalk." He noticed this rock at points more than one hundred and fifty miles distant from each other, and presenting the same lithological characters, and observes that it constitutes the bottom of the many ponds and lakes in the interior of the peninsula; and he was informed that its white and jagged surface can be seen through the whole extent of the Everglades.

That this limestone forms the bottom and rim of the Everglades is confirmed by John W. Newman, C.E., who, in 1892, in company with Mr. Ingraham, then General Manager of the Plant Railway system, made an expedition across the Everglades from Myers, on the Caloosahatchee river, to Miami, on Biscayne Bay.

Dr. T. A. Conrad, a noted and highly-gifted geologist and palæontologist, published observations on the geology of East Florida, in the *American Journal of Science*, in 1846. He describes certain post-Pliocene deposits on the St. Johns river and at Tampa Bay, occurring 10 or 15 feet above high tide, as "proving a considerable elevation of the whole Florida peninsula in the post-Pliocene period, which raised all the Florida keys above water." Conrad also calls attention to an under-

lying limestone, seen at Tampa, Fort Brooke, and Manatee, which he refers to the upper Eocene, and gives a list of the fossils seen, which are characteristic of that epoch.

Professor J. W. Bailey also examined this limestone at many points in Florida, and mentions the fossils found as belonging to the Tertiary.

In 1851, Professor M. Tuomey published an account of the limestone of the keys and the southern coast of Florida. In the limestone at the keys, he found the fossils all identical with the shells lying in the surrounding waters.

At Tampa Bay, he confirms the observations made by Conrad, that the limestone is of Tertiary age; that it extends to Charlotte Harbor, and under the Everglades; and that it is similar in every respect to the limestone at the mouth of the Miami river. Tuomey also states that, "an elevation of the keys of about 10 to 20 feet would produce a ridge," similar to that surrounding the Everglades, "shutting out the sea from the space between the reef and the main land, and producing a second Everglade, differing from the present only in its greater comparative length."

In 1851 Prof. Agassiz and Joseph Le Conte visited Florida to study its keys and reefs. Le Conte considers the inland limestone to be identical with that seen at Vicksburg, and in the southern part of Alabama and Georgia.

Prof. E. A. Smith, State Geologist of Alabama, published in 1881, in the *American Journal of Science* (3d Series, vol. xxi., p. 292), a report on the geology of Florida. He refers the limestone seen in the northern, middle and southern part of the State, excepting the keys, to the Upper Eocene. He considers it identical in age with the Vicksburg limestone, and the limestone seen in the southern part of Mississippi, Alabama and Georgia.

The Upper Eocene character of the inland Florida limestone, as determined by Dr. Smith after a most able and painstaking research, is confirmed by the subsequent examinations made by Prof. E. W. Hilgard, former State Geologist of Mississippi.

William Maclure published a geological map of the United States in 1812. It is printed in colors. The whole of Florida is laid down as alluvial. I believe Mr. Maclure to have been the first to publish a report on the geology of the United States, and to define the boundaries of the formations on a map.

The geological map published in 1876, by Blake and Hitchcock, also lays down the whole of Florida as alluvial.

Having briefly called attention to the views of some of the able geological observers, who have visited Florida and studied its formations, I will add a few words, giving my own views on the subject.

To me the geological history of the peninsula of Florida is plainly written and easily understood. The entire peninsula is underlain by a soft, easily-weathered limestone, which may be designated as the backbone of the State. In many places this rock is so soft and friable that it might well be classed as chalk.* In places it is highly charged with fossils. Palæontologists who have made a study of these fossils refer them to the Upper Eocene age. While agreeing with them, as to the age of this formation, the point to which I wish to call attention does not bind me to any special epoch, as the age of this backbone rock of Florida. It is simply necessary to state, that, in my opinion, it is the oldest rock found in the State. It has an amorphous structure and is of an unknown thickness; wells have penetrated it to a depth of 1200 feet without going through. Locally it contains large and small segregations of chert, an impure flint.

I have traced this limestone from Madison county on the north to Charlotte Harbor on the south, and from Cedar Keys across the State to the St. Johns. Everywhere it presents the same characteristics, sometimes cherty and hard, at other times soft and easily weathered into powder and dropping its fossils, which are mostly in fragments, rarely whole.

It forms the bed of all the numerous fresh-water lakes and ponds, that are so liberally distributed over Florida, as well as the bottom of the St. Johns river, which may be classed as a series of lakes. It is dissolved and worn by water into sinkholes and subterranean caverns through which flow streams of water. At Karlanah, 3 miles north of Williston, in Levy county, a well was drilled to a depth of 130 feet, at which point the bit dropped several feet into a cavity from which an abundant flow of water rose to within 30 feet of the surface. From this well two small living fish were pumped out. One was put into a bucket of water for my inspection, but was so decomposed when I saw it, that the species could not be determined, nor

* Not marl, which is a mixture of clay and carbonate of lime and is of an earthy nature.

could I be satisfied if it was eyeless or not. I am told that at Standard No. 2 phosphate plant, they frequently pump out fish. At the International Phosphate Company's plant, from a well 160 feet deep, I am told they pump out many craw-fish.

The numerous large and beautiful springs of clear water seen in many parts of the State are formed by the weathering of the limestone down to one of these subterranean streams.

Artesian water, nearly always charged with sulphuretted hydrogen gas, is readily found in this limestone along the east and west coasts of the State, at a depth of from 150 to 900 feet. In the central portions of the State, along Trail Ridge, artesian water has not yet been reached. At Ocala and eight miles north of the city, at Anthony, wells were drilled to a depth of 1100 to 1200 feet without finding artesian water.

Florida is not a level plain. Trail Ridge, which is from 30 to 50 miles wide, extends from the North to the Everglades, forming the longitudinal center of the peninsula, and having an elevation of more than 230 feet in places. From this ridge the land slopes in an irregular manner to the Atlantic on the east and the Gulf on the west.

It was not a local but a continental force that elevated the Peninsula of Florida above the water; nor was it of a violent character, but very gradually extended over a vast period of time. The same force, almost simultaneously with the limestone of Florida, brought the tops of the Rocky Mountains above the waters of the Pacific. This continental elevation carried the northern and older portion of North America into a region of perpetual snow and ice, which brought on the glacial epoch that continued in operation, wearing down the mountains and scattering their *débris* over the country to the south. The greatest southern moraine of the glacier is marked by the Ohio river. The elevating of the continent gradually drained the Gulf waters from the land as far north as the junction of the Ohio with the Mississippi river. The glacier continued in action, bringing material from a higher to a lower level until it wore itself out by cutting down its mountain home. It formed the drainage system of the Mississippi river. Almost every succeeding boundary of its receding moraine is well marked in Indiana.* The slow rising of the continent by long intermit-

* See *Geological Report of Indiana*, 1878, by E. T. Cox.

tent stages will account for the laying down of the stratified rocks of North America, from the oldest to the latest epoch.

The Eocene limestone in Florida is filled, for the most part, with fossil marine shells. Conspicuous among them is the characteristic *Orbitoides mantelli.* Fossil coral is rarely seen. I have never found a *Lingula*, nor have I ever heard of one being found.*

Nowhere in the State does this limestone formation show any evidence of disturbance. It presents a solid almost homogeneous mass, without the slightest mark of being stratified. All of its fantastic jagged features are the result of degradation.

Resting on the Eocene limestone are found beds of phosphate of lime. The phosphate does not occur in a continuous layer like seams of coal, but is in detached masses, scattered over an area about 20 miles wide, and extending in a belt that follows in a general way the trend of the Gulf coast from the northern limits of the State and beyond to the western edge of the Everglades on the south. I am aware of small isolated deposits still farther east.

After my first visit to Florida, made early in the spring of 1890, for the purpose of examining some phosphate properties 16 miles west of Ocala, in Marion county, I read a paper at the August meeting in Indianapolis of the American Association for the Advancement of Science, giving my views on the origin and character of the phosphate-deposits in Florida, as well as describing the physical appearance of the mineral. Again, at the Washington meeting of the above association, in August, 1891, I read a second paper, describing the land- and river-pebble deposits, in the district of which Bartow, in Polk county, is the business center. In this paper, for the sake of euphony, I suggested the name of Floridalite for Floridaite, given to this mineral in my first paper. For some reason this name, though simple and suggestive, given to one of the most remarkable and peculiar deposits of phosphate of lime now known to the world, has not been generally adopted.

* I call attention to this fact since some scientists have ascribed the origin of the phosphate deposits in Florida to this shell. The late Dr. Hunt found the shell of *Lingula* to contain 85.79 per cent. of phosphate of lime. Oyster shells contain only 0.5 per cent., and some other shells even less.

Since 1890, I have spent the greater part of my time in studying the phosphate lands in Florida. My explorations have been extended from Madison county in the north to Punta Gorda on Charlotte Harbor in the south. From this extended study I have found no evidence to change my views regarding the origin of the phosphate as derived from the mineralization of an ancient guano.

If the limestone had been phosphatized by the leachings from guano or other sources of phosphoric acid, as suggested by some scientists, we should expect to find the mineral phosphate filled with fossil shells, found so abundant in the limestone, and the mineral in a more compact form. Again, the phosphate, if so formed, could not exhibit such a uniform percentage of phosphoric acid, but would contain less and less as we descended to the bottom of the deposit. The limestone on which the phosphate rests never contains much more than a trace of phosphoric acid, an amount that may be found in almost any rock.

If the phosphate of lime came from shells of *Lingula*, we should surely find at least some of their remains preserved in a fossil state, yet so far the evidence stands against the finding of a single specimen.

With the evidence before us of causes now in action that produced the immense deposits of guano on the islands off the rainless coast of Chili and Peru and the islands of the Caribbean sea and elsewhere, where the droppings of numberless birds are converted into guano, both pulverulent and rock-guano, it is not necessary to construct strange theories to account for the phosphate-deposits in Florida. Unlike Peru, the climate here was humid, and washed out of the guano the soluble salts, leaving the insoluble phosphate of lime.

The isolation of the deposits, their occurrence in detached pockets of greater or less extent, as well as the conformability of the phosphate-rock to the very jagged surface of the limestone on which it rests, all point to its origin as guano.

In the Anthony district, 8 miles north of Ocala, the limestone has been weathered into a remarkable series of jagged and cone-shaped points, and also cut into by numerous pot-holes. Here the phosphate fills the spaces between the cones and irregular points as well as the pot-holes. There is a pot-

hole on the Plate Rock Phosphate Company's (now French Company's) land, that is 6 feet or more in diameter, almost round, gradually tapering to a depth of 40 feet, that was filled with plate-rock and pebble-phosphate. The walls, after the phosphate was taken out, show marks of obsolete rings that indicate a varying rate of cutting down, due probably to storms and high tides acting upon the enclosed stone that did the grinding. Sink-holes are also numerous in the vicinity. Resting on the phosphate and the limestone is a bed of sand that varies from a few inches to 20 feet and more in depth. This covering of sand is found all over the peninsula. It has been blown by the winds from the gulf and ocean beaches; its well-rounded edges and absence of gravel or other coarse material indicate this origin. Mixed with the sand is clay, in the form of fine dust, except where made plastic by the infiltration or uprising of ferruginous water. By the cementing action of chalybeate water, as well as water that contains carbonate of lime, the sand is locally converted into a rock. Sand-rock, like the sand when present, is always found above the phosphate or mixed with it, owing to the sand having filled up the interstices that existed in the phosphate caused by the chemical breaking up of the mass.

At several localities in the State the associated clay is as white as snow. When this clay is separated from the sand by running water it is called kaolin, and is of a very superior quality for the manufacture of the finest quality of porcelain. For this purpose it has been tested in the government experimental pottery at Sèvres, near Paris, France. Here samples taken from Palatlakaha, in Lake county, were tested to determine its chemical constitution, its practical working under the potter's manipulation and its behavior in the kiln, when subjected to the most intense firing. Under each and every treatment at Sèvres it proved to be equal to any other clay known. This kaolin has also been tested in this country by the Trenton and East Liverpool potters; and at each place it gave perfect satistion. It may be worked without the use of "ball-clay," flint, or spar. From 50 to 75 per cent. of the mined material is white sand, which has also been tested at the glass factories in France, where it was pronounced equal to the Fontainbleau sand, so highly prized for making plate-glass.

In Lake county the kaolin bed lies along and contiguous to Palatlakaha creek for nearly its whole length, some 30 miles or more. This creek runs on a high ridge, and the hills near to its head are some of the highest in the State.

For many miles along its course this stream has cut its bed down into the kaolin, leaving both banks as bluffs of fine white clay which, for 25 feet or more, has been in this way opened to view. It is my opinion that the kaolin, with its admixture of sand and fine scales of mica, was washed out from the elevated sand hills by a broad stream of water that flowed from or near the head of the Palatlakaha creek and along its present course. This broad sheet of water had the limestone for its bed, and the kaolin and sand washed from the highland was deposited in the same manner as by the water-process now used to prepare the kaolin for market. The Eocene limestone, with its characteristic fossils, may be seen at many places along the creek.

The Albion Phosphate-District.

BY E. T. COX, ALBION, FLORIDA.

(Florida Meeting, March, 1895.)

THE Albion phosphate-district embraces a territory about 4 miles wide and 6 miles long, in a northerly and southerly direction, situated in Levy county, Florida.

The Florida Central and Peninsular railroad, running from Fernandina, a shipping port on the Atlantic, to Cedar Keys, on the Gulf of Mexico, passes through the center of the district. The post-office and railroad station is in the town of Albion.

The physical features of the district present a series of white and whitish-gray sand-hills and ridges, rising about 130 feet above tide-water. The sand lies in a loose body, like that seen on the ocean beaches. In places the hills have the appearance of sand-dunes, such as are seen along the Platte river, in Nebraska. Beneath the sand are found the phosphate-deposits, which are partially exposed in places, but, for the most part, lie buried to a depth of 15 feet and more.

In this district the phosphate is designated as "hard rock,"

or boulder-phosphate and gravel-phosphate. The boulders range from lumps as large as a man's head up to 50 tons and more in weight. When cleaned, it analyzes from 75 to 85 per cent. of tricalcium phosphate of lime, with a maximum of 2.5 to 3 per cent. of phosphate of iron and alumina. The high grade and excellence of the phosphate of this district have given it a high standing in the market.

The thickness of the deposit runs from 30 to 70 feet and more; there are but few places where mining has gone to the bottom.

The water, which is reached at a depth of from 10 to 20 feet in the phosphate, formerly put a stop to mining. Large pumps were tried, but they could make no impression on the inflow of water. It was said to be like trying to pump out the ocean. On this account it was thought for a time that, after mining down to that level, work would have to be abandoned. Fortunately for the district, Mr. W. N. Camp, of the Camp Phosphate Company, whose mine is about one-third of a mile north of Albion, concluded to try mining with a steam-dredge. Accordingly, he had a dredge-boat made and launched into the pool of mine water. Contrary to all predictions of failure, it proved to be a grand success, and solved the problem of mining the phosphate beneath the water. The importance of being able to mine with a steam scoop-dredge can readily be understood when it is considered that fully two-thirds of the mineral lies below the level of the water.

It is a difficult thing to mine phosphate with pick and shovel, at a cost of $2.50 a ton, where the conditions are most favorable; it will, in my opinion, cost over $3. With a good dredge the cost will rarely exceed $1 a ton.

The Camp Phosphate Co.—This company is mining an average of about 30 tons per day of ten hours. Sometimes it turns out as high as 100 tons. Excavation is carried to a depth of 50 feet below the surface without going to the bottom of the phosphate. The present dredge will go no deeper, but the company is getting a dredge built that will work to a greater depth. The company has taken out of a space about equal to 1 acre as much as 20,000 tons.

Camp Brothers own about 8000 acres of phosphate-land in Albion and High Springs districts. In the latter district they

have three plants running; with these and the Albion plant they average, yearly, an output of 30,000 tons. After spending years in studying phosphate-deposits in Florida, they have recently paid $700 an acre for additional phosphate-land in the Albion district, upon which they are engaged in putting up another plant, with many labor-saving improvements.

At present there is only one other plant using a steam-dredge, namely, that of the Albion Phosphate Mining Co. This company is owned and operated by Baltimore parties, under the management and presidency of Angus Cameron, of Baltimore, Md. It owns or controls a large area of selected phosphate-property, and has a complete plant and mining village within sight of the railroad depot, besides a remarkably good plant, now running, about 2 miles from the Albion plant, with which it is connected by the company's railroad. At this new plant the phosphate is mined by means of a powerful steam-dredge erected on a staunch boat, which floats in a small lake, in which it is moved about.

The crude phosphate is brought up from under the water in a large scoop and emptied into iron cars of special construction which stand on a track supported by a pontoon bridge, and is hauled from the pontoon up an incline and dumped into the log-washer; thence through a cylinder rinser and over the picking-belt, which drops the cleaned rock into cars ready for transportation to the drying-sheds. The operation practically illustrates the adaptation of machinery to dispense with hand-labor, both in mining and in the subsequent preparation of the material for shipment; consequently, there is a great reduction in cost, over the usual way of mining, where water cannot be reached in sufficient quantity to float a dredge. Mr. Cameron declines to give his own estimates of his mining and other costs, but offers every opportunity and facility for observation and individual deductions. It is quite clear to even an inexperienced observer that, in comparison with mining by pick and shovel, the smaller number of hands employed, and the greater volume of material handled by the dredge, undoubtedly constitute an advantage of commercial importance.

The present daily output at this place is said to be 50 tons, which it is intended to increase by further improvements on the plant.

Portland Chemical and Phosphate Co.—The phosphate-property of this company embraces 1500 acres, lying in the center of the district and on the Florida Central and Peninsular railroad. There are two long spurs from this road that run well up into the property, to furnish transportation, on which to locate several plants.

This property adjoins that of the Development Co. on the north, the Camp Co. and Peninsular Co. on the east, the Albion Phosphate Co. on the south, and the Osceola Co. on the west.

It is one of the best developed and most valuable properties in the State.

There are numerous large pits, some of which expose an area of several acres. The bottoms of all the pits are covered with large boulders. Between the pits the ground has been tested by rodding, and marked with stakes that show the presence and continuation of the rock-phosphate; indeed, there are but few places on this property where a pit can be sunk without reaching phosphate, which in quality has no superior in the State. The over-burden is loose sand, easily removed with scrapers. Very little mining for the sale of phosphate has yet been done. There is a plant ready for the arrangement of machinery to clean and prepare the rock for market, and work has been commenced on a plant to manufacture several grades of phosphate-fertilizers, for which business the property is admirably suited, since the small gravel and soft phosphate are remarkably white and of high grade. As yet, work has mostly been done to expose the hard-rock phosphate and point out where plants may be erected when steam-dredges have been procured. In places the bed of phosphate has been proved to a depth of 30 to 40 feet without reaching the bottom (on account of water).

The well-water in this district is abundant, soft and excellent for drinking and domestic use.

The Development Phosphate Co.—This company has a fine property, on which there is a good plant. Mining has been done over a considerable space down to water, and has been temporarily stopped to make arrangements to put in a steam-dredge.

The Peninsular Co.—This Company has a good plant and fine property. It mined and shipped a few hundred tons, but shut

down when water was reached. The plant has been lying idle for about two years.

The Osceola Phosphate Co.—The property of this company lies on the west side of the Portland and about three-quarters of a mile from Albion station. The railroad passes nearly through the centre of the deposit. The business is under the management of William Kissam, of New York City. There is a good plant in operation on the property, and mining is done with pick and shovel. The deposit is very extensive, but the bottom has never been reached. The rock sent to market analyzed over 75 per cent. of phosphate of lime and under 3 per cent. of iron and alumina. The daily output is from 20 to 30 tons. It is the intention of the company to put in a dredge as soon as sufficient mining has been done to give room for a boat.

Gen. E. B. Bailey's Property.—This property lies about 2 miles southeast of Albion station. Gen. Bailey employs convict labor, and has about 140 men mining. He commenced work on this property in the autumn of 1894. A large amount of rock, of excellent grade, has been mined and piled up ready for shipment as soon as the spur—1¾ miles long—from the main railroad line is completed, which will be in a few days (March, 1895).

Here, as on the Portland property, the boulders are remarkably large and of high grade. Gen. Bailey also will soon be compelled to use a dredge to mine below the water-level.

Florida has the only high-grade phosphate now known to the world that can be had in any quantity desired. The prevailing low price has prevented a more rapid development of the Albion district, some holders of land preferring to leave the rock in the ground rather than sell at present prices, in view of the probable future advance.

North Carolina Monazite.

BY H. B. C. NITZE, BALTIMORE, MD.

(Florida Meeting, March, 1895.)

MONAZITE is a phosphate of the rare earths, cerium, lanthanum, and didymium (Ce, La, Di)PO_4. It also contains thoria (ThO_2) and silica, which are present in varying percentages, probably as impurities.

It is a subtranslucent to transparent mineral, light-yellow, yellowish-brown, or yellowish-green in color, and has a resinous lustre. Its hardness is from 5 to 5.5, and its specific gravity from 4.9 to 5.3. It crystallizes in the monoclinic system, and the crystals are usually small.*

The economic value of monazite lies principally in the thoria which it contains; this is used as one of the constituents in the manufacture of the mantles for the Welsbach and other incandescent gas-lights.

Monazite is somewhat widely distributed, but until now has been found in commercial quantities only in Brazil (Minas Geraes, Caravellas, and Bahia), Siberia, Norway, and in the States of North and South Carolina. The North Carolina area embraces between 1600 and 2000 square miles, situated in Burke, McDowell, Rutherford, and Cleveland counties. The principal deposits of this region are found along the waters of Silver, South Muddy, and North Muddy creeks, and Henry's and Jacob's Forks of the Catawba river in McDowell and Burke counties; the Second Broad river in McDowell and Rutherford counties; and the First Broad river in Rutherford and Cleveland counties. These streams have their source in the South Mountains.

Some monazite is also found in Polk county, and along the western edge of Catawba, Lincoln, and Gaston counties, but, so far as present explorations go, the quantity is small and the quality inferior. The mineral is also reported as occurring in Madison, Mitchell, and Yancey counties, but not in commercial quantities. The best crystallized specimens have been found at Milholland's mill and Stony Point in Alexander county. The monazite occurs in the sands and gravels of the stream beds, associated with other minerals, such as quartz, feldspar, hornblende, epidote, mica, magnetite, garnet, zircon, rutile, corundum, etc. It is separated from most of these, owing to its superior specific gravity, by washing in sluice boxes in much the same way that placer gold is won. Magnetite can be eliminated by treating the dry sand, after washing, with a magnet.

The primary source of monazite, as of many of the other rare minerals in this region (such as zircon, xenotime, ferguson-

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* For further crystallographic and mineralogic notes, see Dana's *System of Mineralogy*, 6th ed., 1892, pp. 749–752.

ite, etc.), is in the crystalline gneisses and schists, of which it is an accessory constituent. The material produced from the disintegration of the decomposed country rock is deposited in the stream beds, and undergoes, by virtue of a continual current and differences of specific gravity, a natural process of partial sorting and concentration. The richer portions of the stream deposits are thus, as a rule, found near the head-waters.

As the percentage of thoria is variable in different sands, the value of the mineral consequently varies accordingly, and must be determined by careful chemical analysis. Some monazite contains practically no thoria. It is stated that the transparent greenish and yellowish-brown varieties are usually the richest.

The best North Carolina sands (highest in thoria) occur near Brindletown, Burke county, and in the northern part of Cleveland county. Some of the highest grade Brindletown sand runs from 4.00 to 6.60 per cent. of thoria; sand from Gum Branch in McDowell county is reported to run 3.30 per cent. thoria; some sand from near Shelley in Cleveland county contains 2.76 per cent. thoria.

The fluctuations in the thoria constituent are, however, considerable, even in the same locality. It also depends of course in a measure on the concentration of the monazite in the cleaned sand; many of the heavy minerals, such as garnet, zircon, menaccanite, rutile, corundum, etc., cannot be perfectly eliminated. The commercially prepared sand, therefore, after washing over several times and treating with a magnet, is not *pure* monazite. A cleaned sand containing 60 per cent. monazite is considered good.

The thickness of these stream-gravel deposits is from 1 to 2 feet, and the width of the mountain streams in which they occur is seldom over 12 feet. The sluice boxes are about 8 feet long by 20 inches wide by 20 inches deep. Two men usually work at a box, the one digging the gravel and shovelling it into the box, the other one working it up and down in the box with a gravel-fork or perforated shovel in order to float off the lighter sands.

These boxes are cleaned out at the end of the day's work, the cleaned monazite being collected and dried. If it contains magnetite it is treated with a magnet. It is then ready for

packing and shipment. From 20 to 35 pounds of cleaned monazite sand, per hand, is considered a good day's work. The value of the best grades of sand is 6 to 7 cents per pound at the diggings.

During the past two years the following shipments of monazite sand have been made from this region:

In 1893:

Pounds.	Value at mines.
110,000 @ 6 c.,	$6,600 00
20,000 @ 5 c.,	1,000 00
Total, 130,000,	$7,600 00

In 1894:

Pounds.	Value at mines.
460,000 @ 6¾ c.,	$31,050 00
80,000 @ 6 c.,	4,800 00
6,855 @ 5 c.,	342 75
Total, 546,855,	$36,192 75

A Water-Cooling Apparatus.

BY CARL HENRICH, DUCKTOWN, TENN.

(Florida Meeting, March, 1895.)

In the planning and erection of smelting-works, especially of such as contain the modern large water-jacketed blast-furnaces, we are often confronted with an insufficiency in the water-supply. It may be impossible to find in the immediate neighborhood of the selected site a sufficient supply to furnish the jackets with cold water. Or there may be enough water for this purpose, but it may be highly charged with scale-forming minerals held in solution. Or, the main water-supply, if pumped from the workings of a mine, may carry iron- or copper-salts, or even some free acid in solution, all of these substances being derivable from the oxidation of sulphuret-ores in the mine. Moreover, such mine-waters sometimes contaminate the waters of the creeks, so that these become unavailable, and we have to fall back on an inadequate supply of pure water furnished by some spring or springs, entirely too small in

volume for our needs, unless used over and over again. This last alternative, using the same water repeatedly for the cooling of the water-jackets of a furnace, necessitates, of course, the cooling of the heated water before each new introduction of it into the water-jackets. The effective cooling of the large volumes of water needed, even within the space of a few hours, for the water-jackets of the modern large blast-furnaces, presents the greatest difficulty attendant upon the repeated use of the same water for this purpose.

The writer has seen in the West attempts to solve this problem by employing a series of large shallow wooden tanks. Connection between these tanks was, or ought to have been, made in such a manner that the bottom of one tank would communicate with the top of the next, the hot water entering at the surface at one end of the series, and the cooled water being taken from the bottom of the tank at the other end. But to cool any large quantity of water in this way requires, even in the arid regions of the west, where evaporation is much more effective than is usually the case elsewhere, so large a number of large tanks, costly to build, and costly to keep in effective repair, that this plan cannot be called a success.

Recently, in planning and building the smelting-plant of the Pittsburgh and Tennessee Copper Company, at Ducktown, Tennessee, the writer was confronted with a problem of the kind mentioned above.

The country around Ducktown is well watered, and would have seemed, at first sight, the most unlikely region to present such a question. But the drainage of the old mine-openings (mostly caved in) empties into the creeks of the region; and these mine-waters are so much impregnated with the sulphates of iron (and probably also a little free sulphuric acid) as to make them very undesirable for use in boilers and iron water-jackets. On the other hand, the springs (which are frequent, but not, as a rule, very large) furnish a water almost as soft as rain-water, and forming no scale worth speaking of; in other words, an ideal material for such purposes.

A group of springs, immediately below the site selected for the works, yields ordinarily from 35 to 40 gallons of water per minute. In the abnormally long-continued drought prevailing in the region in 1893 and 1894, this supply fell to about 25 gal-

lons per minute, which was taken as the amount to be safely depended on at all times. Arrangements were therefore to be made to make this supply of 25 to 30 gallons per minute sufficient for the needs of the works.

Of the furnace, for the water-jackets of which this supply had to be sufficient, the horizontal area at the tuyere-level was 120 by 42 inches; the area at the top of the jackets (which was also the bottom of the charge-doors), 126 by 54 inches; the total height of the jackets, 12 feet 5 inches. Sixteen 4-inch tuyeres supplied the blast. Besides the jackets of this furnace, a small matte-concentrating cupola, 32 to 36 inches in diameter at the tuyeres, was to have been supplied with water from the same source.

The water for the boilers could be neglected, as this would be taken from the hot water coming from the furnace, and would be a factor only in the determination of the size of the pump required for pumping the hot water, as it came from the jackets, to the cooling-apparatus.

An Epping plunger-pump, No. 7, with outside-packed plunger, 12-inch steam cylinder, 7-inch plunger, and 14-inch stroke was the compromise between the real requirements of the case and the least possible expenditure desired. It would have been much better to increase the size of that pump to at least No. 8, with 14-inch steam cylinder, 8-inch plunger, and 14-inch stroke; and a No. 10 pump of the same make, is what the writer would recommend in a similar case. The No. 7 pump is rated at 200 gallons per minute with 100 feet piston-speed; but this speed was often exceeded, when running the furnace to anything like its full capacity—about 240 tons of roasted pyrrhotite-ore in twenty-four hours.

The main problem remains, namely, the cooling of 200 to 300 gallons per minute of water heated to about 180° to 200° F., down to a temperature as near that of the spring-water (about 40° F.) as possible. For this purpose the writer designed the simple apparatus here described. While convinced that it would serve its purpose, still he hardly supposed that the small size of the apparatus first built would be sufficient, and fully expected to have to add 50 to 100 per cent. in length, or to double the original apparatus, in order to obtain the water sufficiently cooled. But the efficiency of the apparatus in

actual practice has proved so great that he feels justified in publishing it for the benefit of others in the profession similarly situated.

The apparatus is shown in vertical cross-section in Fig. 1, and the arrangement of the distributing strips, which are the essential portion, is shown in Fig. 2 (a part of the vertical cross-section on an enlarged scale).

In this description I shall give the dimensions used in building the cooler in actual use, adding, however, such suggestions of improvement in its construction as experience has shown to be desirable. The apparatus is made of wood throughout. Any kind of wood may be employed which will last in water, and can stand repeated dryings-out (in times when the works are idle) and soakings in hot water (when the works are running).

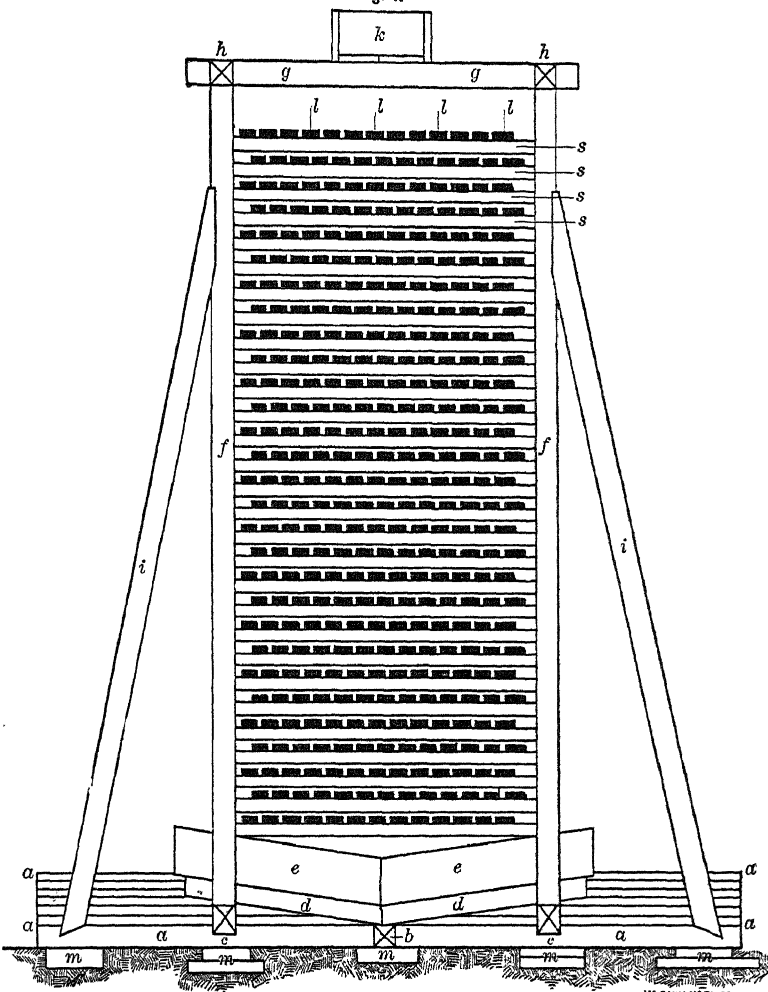
The construction is based on the well-known fact that for a given volume of hot water, other conditions being equal, the velocity of cooling will increase with the increase of the surface brought into contact with the air. Of course, this cooling is effected mainly through the evaporation of a portion of the water, the heat rendered latent by such evaporation being taken from the remaining portion of the water, while the stream of air moving constantly over the surface of the water serves, in the main, only as the vehicle for carrying off the evaporated water.

A similar idea was put into practice long ago at some of the salt works of Europe, where the writer saw it in use, in his boyhood, for concentrating the weak brine pumped from the salt-mines. In these salt-works the brine was pumped to the top of long, high, but narrow piles of bundles of willow twigs, tied together, and confined within a light wooden frame-work, the whole being exposed to the currents of the air, which passed freely through this pile of twigs, while the finely-divided brine dropped from one twig to the other, to be collected in a concentrated state on the floor on which the whole structure rested, and thence flowed to the pans, in which it was boiled down.

In the Ducktown apparatus, modified from this pattern, the frame-work was retained, but the bundles of twigs were replaced with long, narrow, thin wooden strips (set perfectly level) on which the hot water would spread out, and over the

edge of which it would drop upon other strips, put vertically below the interstices left between the layer of parallel strips above. A V-shaped floor was put below this arrangement of strips, to collect the cooled water and deliver it, by means of a

Fig. 1.



WATER COOLER. VERTICAL CROSS-SECTION
Scale: ¼ in. 1 ft.

launder, to the two large tanks, which were to form the reservoir supplying cold water for the smelter.

As constructed, the frame-work of the apparatus consists of the cross-sills, *a*, *a*, Fig. 1, set parallel and level, 8 feet apart from center to center. Each cross-sill was put 2 inches higher than

the one in front of it, so that the seventh cross-sill, at the end of the frame-work, is just one foot higher than the first cross-sill in front. These cross-sills are laid on mud-sills, *m*, *m*, which are made of 2- by 12-inch planks, firmly bedded in the ground.

The cross-sills, *a*, *a*, are joined in the center by a long sill, *b*, halved into the cross-sills, so that their tops flush. Besides this center long sill, *b*, two other long sills, *c*, *c*, join the cross-sills, *a*. These outer sills, *c*, are equidistant from the center sill, *b*, and parallel to it and to each other. They are 7 feet apart at their inside edges, and being made of 6- by 6-inch lumber, are 8 feet apart from outside to outside. The center sill, *b*, is made of the same size (6- by 6-inch) while the cross-sills *a* are made of 4- by 6-inch stuff, put on edge. The outer sills, *c*, are let into the cross-sills only 2 inches, so that the top of these outer sills is 4 inches higher than the top of the center sill, *b*.

On the intersections of the outer sills, *c*, with the cross-sills, *a*, posts, *f*, from 16 feet 6 inches to 17 feet 6 inches in height, made of 4- by 6-inch lumber, are erected. They are braced to the cross-sills, *a*, by the braces, *i*, made of 4- by 4-inch lumber.

Each pair of posts is joined on top by the cross cap, *g*, made of 4- by 6-inch lumber; and the 7 pairs of posts are joined on top by the two long caps, *h*, made of the same size. Instead of putting the cross-sills, *a*, and posts, *f*, 8 feet apart from center to center, it would have been better to diminish that distance. By using 4 feet of distance, *i.e.*, 13 cross-sills, *a*, 13 pairs of posts, *f*, and 13 cross caps, *g*, instead of 7, it would have been possible to put the distributing strips, *l*, more nearly level, than with supports 8 feet apart. Besides the braces, *i*, X-braces, made of 3- by 6-inch lumber, are used on the long sides of the frame-work between the posts, to stiffen the structure.

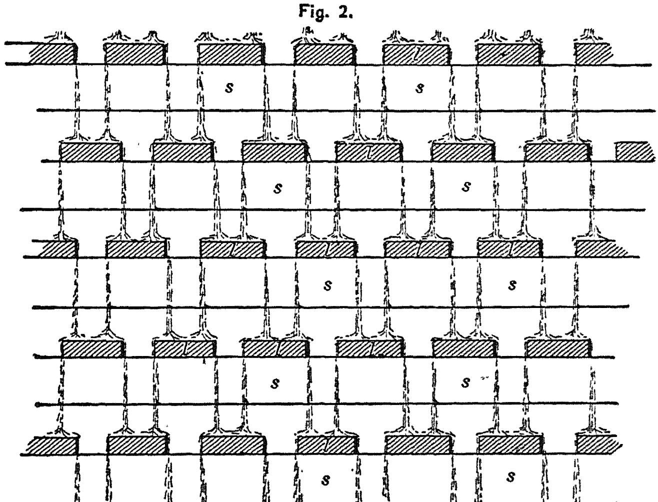
Between each pair of posts, *f*, *f*, strips of wood, *s*, *s*, 2- by 4-inch, and 7 feet long, are nailed, 6 inches apart from center to center, perfectly level, and so placed that the distributing strips, *l*, when laid on these supports, *s*, lie perfectly level in the direction of the length of the structure.

The distributing strips, *l*, are battens 1 by 4 inches in size, and 16 feet long, reaching over the length of two panels of the structure, and breaking joints, which adds materially to the stiffness of the whole.

The correct placing of these distributing strips, *l*, insures the

efficiency of the apparatus. Each of these should be perfectly level in both directions (the length and breadth of the structure). Their correct position to each other is shown in Fig. 2, an enlarged vertical cross-section through a part of the center of a panel of the structure. In this figure the distributing strips, *l*, are shown 4 inches wide, by 1 inch thick, and lying parallel, with interstices 2 inches wide, in layers 6 inches apart vertically, from center to center. The water falls over the edge of each strip upon the upper face of a strip immediately below. It spreads out on this level surface or is thrown up, in drops or

Fig. 2.



VERTICAL CROSS-SECTION, (PART OF FIG. 1. ENLARGED) SHOWING RELATIVE POSITION OF DISTRIBUTING-STRIPS
Scale 1″=1 Ft.

spray to be collected on neighboring strips, over the edges of which it falls again, and so on. The nearer to a perfect level the upper faces of these strips are brought, the more perfect will be the spreading out and distribution of the water over the whole apparatus.

Of course this uniform distribution of the water depends also on the arrangement of the holes or slits in the upper trough or launder (*k*, Fig. 1) of the apparatus, which is also perfectly level, and into which the water flows from the 4-inch discharge-pipe of the hot-water pump. The uniform distribution of the water from this launder, *k*, over the whole length and breadth of the apparatus can be easily accomplished, even after the apparatus

is at work, by boring auger-holes through the bottom and sides of the launder (made of 1½-inch planks).

In practice it has been found advisable to put up a wind-break at one side of the apparatus, since, without this protection, any strong wind is liable to carry a large portion of the water in the form of a spray (a perfect rain-shower) to one side, beyond the floor of the apparatus. This wind-break was made of rough 1-inch planks nailed horizontally to the braces, *i*, outside from the floor about two-thirds up their length; the upper edge of this lower part of the wind-break being then connected by vertical 1-inch planks with the long cap, *h*, at the top of the apparatus.

The lumber-bill for the apparatus described, putting cross-sills, *a*, and posts, *f*, 4 feet apart (in place of 8 feet as constructed), would be as follows:

(*m*) Mud-sills (about), .	26 pieces	2	by	12 in.	by	10 ft.	=	520 ft.		
(*a*) Cross-sills, . .	13 "	4	"	6 "	"	16 "	=	416 "		
(*b* and *c*) Long sills, . .	9 "	6	"	6 "	"	18 "	=	486 "		
(*f*) Posts, . . .	26 "	4	"	6 "	"	18 "	=	936 "		
(*i*) Braces, . . .	26 "	4	"	4 "	"	16 "	=	554 "	8 in.	
(*g*) Cross-caps, . .	7 "	4	"	6 "	"	18 "	=	252 "		
(*h*) Long-caps, . .	6 "	4	"	6 "	"	18 "	=	216 "		
x-bracing at sides . {	36 "	3	"	6 "	"	18 "	=	972 "		
(not shown in Fig. 1), {	16 "	2	"	6 "	"	12 "	=	192 "		
(*d*) Joists, . . .	13 "	2	"	4 "	"	10 "	=	86 "	8 "	
(*k*) Launder (24 by 10½″)	12 "	1½	"	12 "	"	18 "	=	324 "		
(*e*) Flooring (⅞ by 4″), .	about .	.	.	.	.	.	.	600 "		
(*s*) Supports, . . .	195 pieces	2	by	4 "	"	14 "	=	1,820 "		
(*l*) Distributing strips, .	1170 "	1	"	4 "	"	16 "	=	6,240 "		
Wind-break, . . . {	36 "	1	"	12 "	"	16 "	=	576 "		
	24 "	1	"	12 "	"	14 "	=	336 "		
Total, ...								14,527 "	4 "	

Counting the cost of lumber (inclusive of nails and cost of erection) at $20 per thousand feet, board-measure, and allowing $5 for painting the floor (which was, in the case described, done with roofing-paint), and $15 for grading and preparing the site, we have as the total cost of the apparatus $310.54, which is certainly as cheap as any apparatus capable of efficiently cooling about 200 gallons of nearly boiling water per minute has ever been built, according to the writer's experience and judgment. The simplicity and cheapness of this efficient apparatus, which may be built by a common carpenter anywhere, seem to justify its description in the *Transactions*.

Nickel and Nickel-Steel.

BY FRANCIS L. SPERRY, CANADIAN COPPER CO., CLEVELAND, O.

(Florida Meeting, March, 1895.)

UP to within a few years, the consumption of nickel has been more directly dependent upon the available supply than that of any of the other useful metals.

The Gap mine, in Lancaster county, Pennsylvania,* has been, for the last quarter of a century, the only property in this country furnishing nickel in paying quantities. Its yearly output was about 300,000 pounds of metallic nickel, or nearly half the amount used annually in the United States. Foreign nickel from mines on the New Caledonia islands, in the South Pacific, found entrance into our markets as the production of the Gap mine fell off. The price of nickel was constantly maintained, and no special effort was made to extend its use. Over-production was cautiously guarded against, and all surplus metal was held by the banking-houses of the Rothschilds, who assumed the bonded indebtedness of the *Société le Nickel.* The opening of the Ontario nickel-mines has, however, brought about a radical change; and nickel from the Sudbury district can be delivered in New York within four days, and in European markets within two weeks, as against two months consumed in transporting South Pacific ores. Former prices have been irretrievably smashed, and European trade-journals comment favorably on the influence which Canadian nickel has had in making lower prices, and breaking the backbone of the "nickel trust."

PRODUCTION AND COSTS.

The quantity of nickel produced and the prices which it commanded may be briefly summarized as follows:

The total production of the world from 1840 to 1860 was about 100 to 250 tons yearly of metallic nickel; from 1860 to

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* See *Trans.*, xxiv., 620, 883.

1870, 600 to 700 tons yearly; from 1870 to 1889, about 1500 tons yearly; in 1890, 2000 tons; and a fair estimate for 1894 is about 5000 tons. The metal sold for $2.25 per pound in 1860; in 1873 to 1875, for $6 to $7 per pound. From that time the price of nickel gradually declined, being $0.65 per pound in 1892, and less than $0.40 at the present time. The exceedingly high prices in 1873 to 1875 were caused by the adoption of a nickel coinage by Germany and some other European nations, causing a sudden demand which exceeded the supply.*

Properties of Nickel.

Nickel has physical properties similar to those of iron and copper. It is less malleable and ductile than iron, and less malleable and more ductile than copper. It alloys with these metals in all proportions. It has nearly the same specific gravity as copper, and is slightly heavier than iron. It melts at a temperature of about 2900° to 3200° Fahr. A small percentage of carbon in metallic nickel lowers its melting-point perceptibly. Nickel is harder than either iron or copper; is magnetic, but will not take a temper. It has a grayish-white color, takes a fine polish, and may be rolled easily into thin plates or drawn into wire. It is unappreciably affected by atmospheric action, or by salt water. Commercial nickel is from 98 to 99 per cent. pure. The impurities are iron, copper, silica, sulphur, arsenic, carbon, and (in some nickel) a kernel of unreduced oxide. It is not difficult to cast, and acts like some iron in being cold-short. Cast bars are likely to be porous or spongy, but, after hammering or rolling, are compact and tough. A piece of pure nickel rolled plate (A) and an untreated cast bar of nickel (B) were submitted to physical test by the writer, at the works of the Carbon Iron Co., Pittsburgh, Pa., with the following results:

Cross-section. Inches.	Length between fillets. Inches.	Ultimate strength per sq. in. Pounds.	Reduction of area. Per cent.	Ultimate elongation. Per cent.
A—3.11 by .045	8	69,390	31.5	31.4
B—0.623	2	30,985	6.5	6.5

The following table shows the properties of the metal.

* J. H. L. Vogt, *Nikkel-forekomster og Nikkel-produktion*, Kristiania, 1891.

Test of Strength of Malleable Nickel.

Material.	Tensile Strength. Pounds per square inch.	Elongation. Per cent.	Remarks.
Casting	85,000	12	
Wrought nickel	96,000	14	Wrought from 2 by 4 inches to ½ inch square.
Wrought nickel, annealed	95,000	23	Wrought from 2 by 4 inches to ½ inch square.
Rolled nickel	78,000	10	Very hard, because not annealed after rolling; rolled from 2 to ⅛ inch.

These figures are an average of a number of tests. As there were flaws in several of the specimens, the results are lower than they otherwise would have been.

Nickel readily takes up carbon, and the porous nature of the metal is undoubtedly due to occluded gases. According to Dr. Wedding,* nickel may take up as much as 9 per cent. of carbon, which may exist either as amorphous or as graphitic carbon, or in both conditions. The affinity which nickel shows for carbon is manifested in a striking manner in the Mond process of refining nickel.

Dr. Fleitmann, of Germany, first discovered that the use of a small quantity of pure magnesium would free nickel from occluded gases and give a metal capable of being drawn or rolled perfectly free from blow-holes. Magnesium in nickel, like manganese in steel, acts as a purifying agent, and it improves the ductility and malleability of nickel to such an extent that the metal may be rolled into thin sheets 3 feet in width. Aluminum or manganese may be used equally as well as a purifying agent; but either, if used in excess, serves to make the nickel very much harder.

Nickel-Alloys.

Nickel will alloy with most of the useful metals, and generally adds the qualities of hardness, toughness, and ductility. It is commonly alloyed with copper and zinc in making the composi-

* *Stahl u. Eisen*, No. 8, 1893, p. 328.

tion known to the trade as German silver, white metal, British plate, Packfong or Chinese metal, Argentan, Electrum, and Maillechort, the hardness and whiteness of this alloy depending upon the percentage of nickel it contains. Nickel coins current in Germany, Belgium, Italy, the United States, and Latin American countries, contain 25 per cent. of nickel and 75 of copper. German silver has a considerable use in electrical fixtures and appliances, having a very high specific resistance.

The alloy known as "Christofle" is composed of 50 parts nickel and 50 parts copper. As yet comparatively little use is made of this alloy in the United States; abroad, it is largely employed in the manufacture of coachmakers' and saddlers' supplies, as well as for surgical instruments.

Analyses of nickel-alloys of various countries do not show very great difference in the percentage of nickel.

Analyses of Nickel-Alloys.

	Copper. Per cent.	Nickel. Per cent.	Zinc. Per cent.	Iron. Per cent.	Cobalt. Per cent.
Berlin Alloys.					
Richest, ...	52.00	22.00	26.00	......	......
Medium, ...	59.00	11.00	30.00	......	......
Poorest, ...	63.00	6.00	31.00	......	......
French Alloys.					
Tableware, ...	50.00	18.70	31.30	......	......
"	50.00	20.00	30.00	......	......
Maillechort, ...	65.40	16.80	13.40	3.40	......
Austrian Alloys.					
Tableware, ...	50.00	25.00	25.00	......	......
"	55.60	22.20	22.20	......	......
"	60.00	20.00	20.00	......	......
Sheffield, England, Alloys.					
Silver white, ...	55.20	20.70	24.10	......	......
Electrum, ...	51.60	25.80	22.60	......	......
Hard alloy,*	45.70	31.30	20.00	......	......
English, ...	60.00	18.80	17.80	......	3.40
" elastic, ...	57.00	15.00	25.00	......	3.00
Chinese packfong, ...	40.40	31.60	25.40	2.60	......
American Alloys.					
Alloy for castings, ...	52.50	17.70	28.80	......	......
" " bearings, ...	50.00	25.00	25.00	......	......
Bullet-shell, ...	75.50	24.10	......	0.40	......
One-cent coin, ...	88.00	12.00	......	......	......

* Can be worked cold.

		Per cent.
Vivian & Co., Swansea, copper-nickel alloy, . . .	Si,	.303
	Fe,	.826
	Cu,	48.49
	Ni,	50.09
Société le Nickel, Paris, copper-nickel alloy, . .	Si,	.186
	S,	.089
	Cu,	48.740
	Ni,	49.26
	Fe,	.610
Wiggins & Co., Birmingham, England, copper-nickel alloy, ...	Si,	.136
	S,	.041
	Cu,	47.68
	Ni,	49.87
	Fe,	1.228

Steel and Nickel-Steel.

It will hardly be questioned that scientific research is directed most energetically at the present time upon the art of uniting elements in such proportions that they may be more serviceable than in their pure state. The limit of ultimate strength in the practical application of pure metals has about been reached. The practical introduction of steel into general use has made a new era in manufactures, and "steel is only modified iron; the difference in its state from a condition as soft as copper to one as hard as glass being due to the modifications of carbon." Up to recent times the distrust of steel was so great that marine and civil engineers were afraid to use it. In the early days of the Pennsylvania railroad, its steel rails were imported from England, bent to the curves of the roadbed. As a superior metal for cutlery and tools it brought a fancy price of 36 cents per pound. To-day our battle-ships are sheathed with thousands of tons of the best steel, and 800 tons are used yearly in the manufacture of steel pens. Carbon-steel was a great improvement over iron, and the use of nickel in steel is found, in all cases in which careful investigation has been made, to mark a further improvement in the manufacture of steel. A German authority has recently observed that, considering the mutual affinity of nickel and iron, as shown by the presence of nickel in meteoric iron, it is remarkable that the example of the handiwork of Nature had not been copied before this.

In a paper read before the Iron and Steel Institute, Mr. James Riley, manager of the Steel Works of Scotland,* says:

* "Alloys of Nickel and Steel," *Journ. I. and S. Inst.*, No. 1, 1889, p. 54.

"If the engineers of those stupendous structures (the Forth Bridge and the Eiffel Tower) had had at their disposal a metal of 40 tons (ultimate) strength and 28 tons elastic limit, instead of 30 tons strength and 17 tons elastic limit in the one case, and say 22 tons strength and 14 to 16 tons elastic limit in the other, how many difficulties would have been reduced in magnitude as the weight of material was reduced !"

Mr. Riley's paper was the first to present publicly the merits of nickel-steel, and attracted much attention.

Just about that time the Ordnance Bureau of the United States Navy Department was seeking the best type of armor-plate for the new battle-ships, and the superior qualities of nickel-steel were brought to the attention of the department. Secretary Tracy authorized a comparative trial of three armor-plates forged at the largest steel-works in France and England, and representing the best types of simple steel, nickel-steel and compound (hard and soft) steel armor-plates. The result of this trial, in September, 1890, indicated so strongly the superior merits of nickel-steel that Congress was justified in granting an appropriation for the purpose of purchasing the necessary quantity of nickel to continue experiments. These experiments were uniformly successful, and the Navy Department adopted nickel-steel for armor-plate, and, wherever possible, in the work of the Ordnance Bureau. Nickel-steel armor of the best quality is now regularly produced by two of the large steel-works of Pennsylvania, the Bethlehem Iron Co. and the Carnegie Steel Co., which have special facilities for handling this class of work. The former concern forges all its plates, while the latter employs rolls.

The Harvey process of hardening the face of nickel-steel armor by cementation to the depth of several inches, with subsequent water-hardening, is an important advance in making nickel-steel armor still more effective.

The type of armor-plate used by the British Admiralty is a compound plate made up of a hard steel face and soft steel backing. They considered the question of the best armor for their battle-ships as settled in 1878, when they adopted this type of armor-plate. Comparing the relative depth of penetration in the Harveyized nickel-steel, all-steel, compound and soft-steel armor-plates, the ratio of superiority in favor of the Harveyized nickel-steel plate is as follows, in the order named:

Relative penetration.	Kind of armor plate.	Relative resistance.
1.	Nickel-steel, Harveyized.	1.
1.64	All steel.	0.609
1.75	Compound.	0.572
2.2	Soft steel.	0.455

so that for equal power of resistance there can be a saving of 43.8 per cent. in weight, in favor of the Harveyized plate over the compound plate.* The ordnance trials at the Indian Head proving-grounds are as severe as any in the world; and it is with pardonable pride that the Bureau of Ordnance of the Navy regards the placing of an order for nickel-steel armor-plate by the Russian government with the Bethlehem Iron Company as an acknowledgment that we have, to-day, the material and facilities, and are forging in this country armor and projectiles that have no superior in the world.†

Krupp, of Essen, is furnishing, for vessels of the "Brandenburg" class in the German navy, nickel-steel armor made on a new system. The plates are 5½ inches thick, and show a resistance equal to plates of 9¾ inches made by the old system.

The French government uses an armor-plate containing 0.4 per cent. carbon, 1 per cent. chromium and 2 per cent. nickel.

Nickel furnishes toughness; and chromium, hardness. It is in the highly desirable qualities of extreme toughness and elasticity that nickel imparts valuable properties to steel, increasing its resistance to shocks and hindering crystallization.

The Bureau of Steam Engineering, United States Navy, has had the two intermediate line-shafts of the "Iowa" and the two propeller-shafts of the "Brooklyn" made of nickel-steel by the Bethlehem Iron Company. The line-shafts are 15¾ inches outside and 9¾ inches inside diameter, while the propeller-shafts are 17 inches outside and 11 inches inside diameter; the walls being in both cases 3 inches thick. The government specifications require a tensile strength of 85,000 pounds and 50,000 pounds elastic limit. Six test-pieces from one of the propeller-shafts of the "Brooklyn" gave the following results:

Nickel-Steel Propeller-Shaft for U. S. Ship "Brooklyn."

Hollow-forged, oil-tempered, rough-machined. Outside diameter, 17⅛ inches; inside diameter, 11 inches; length, 38 feet 11⅜ inches; weight, 19,112 pounds.

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* See *Stahl u. Eisen*, No. 4, 1893, p. 143.

† See article "British Armor and Ordnance," *London Engineer*, March, 23, 1894.

Test bars cut from this tube gave the following results:

Dimensions of specimens. Inches.	Tensile strength. Lbs. per sq. in.	Elastic limit. Lbs. per sq. in.	Elongation. Per cent.	Contraction. Per cent.	Fracture.
0.496 by 2	94,185	58,995	26.4	60.83	Dense gray lipped.
0.497 by 2	94,245	60,770	25.55	60.58	" " "
"	93,215	58,740	25.8	61.33	" " "
"	93,730.	60,770	25.8	59.81	" " "
0.498 by 2	92,410	59,550	28.0	60.74	" " "
"	90,350	56,470	28.0	60.74	" " "

It is to be noted that the elastic limit of this shaft is about equal to the tensile strength of a shaft made of ordinary mild steel, while the elongation and contraction of area are nearly the same.

A comparison of the strength of the nickel-steel shafts of the U. S. vessels "Brooklyn" and "Iowa," within their elastic limits, with that of solid shafts of the same sectional area made of soft, simple steel, having an elastic limit of 30,000 pounds per square inch, and also a comparison of their weights per linear unit with that of solid soft steel shafts of equal strength, may be of interest. The following table gives the results of calculations made by Prof. Mansfield Merriman, Lehigh University, Pa.:

Case I. Comparison of three steel shafts.	Propeller shaft U. S. S. Brooklyn. Hollow. Outs. diam. 17 inches; ins. diam. 11 inches. Nickel steel, E. L. 50,000 lbs. per sq. in.	Solid shaft, same (approximate) sectional area. Diameter 13 inches. Simple steel, E. L. 30,000 lbs. per sq. in.	Solid shaft, same strength under applied loads or horse-powers. Diameter 18.9') inches. Simple steel, E. L. 30,000 lbs. per sq. in.
Area of section, square inches, ...	131.95	132.73	280.55
Weight per yard, pounds, .	1,346	1,354	2,861
Comparative strength under applied loads in flexure, or under applied horse-power in torsion, . .	307	100	307
Load, in pounds, at middle of a span of 12 feet on two supports, which strains to one-half elastic limit, .	276,200	89,900	276,200
Length of beam on two supports, which is strained by its own weight to one-half elastic limits, . .	121 ft. 6 in.	77 ft. 6 in.	83 ft. 4 in.
Horse-power transmitted at 50 revolutions per minute when strained to one-half elastic limit, . . .	15,780	6,130	15,780

Case II. Comparison of three steel shafts.	Intermediate Line shaft U. S. S. Iowa. Hollow. Outs. diam. 15¾ inches; ins. diam. 9¾ inches. Nickel Steel, E. L. 50,000 lbs. per sq. in.	Solid shaft, same (approximate) sectional area. Diameter 12⅜ inches. Simple steel, E. L. 30,000 lbs. per sq. in.	Solid shaft of same strength under applied loads or horse-powers. Diameter 17.71 inches. Simple steel, E. L. 30,000 lbs. per sq. in.
Area of section, square inches, ...	120.17	120.28	246.34
Weight per yard, pounds, .	1,225	1,227	2,513
Comparative strength under applied loads in flexure, or under applied horse-power in torsion, . .	293	100	293
Load which, at middle of a beam 12 feet in span on two supports, causes strains equal to one-half elastic limit, pounds, ...	227,200	77,500	227,200
Length of beam on two supports which is strained by its own weight to one-half elastic limit, ...	115 ft. 6 in.	75 ft. 9 in.	80 ft. 8 in.
Horse-power transmitted at 50 revolutions per minute when strained to one-half elastic limit, ...	12,980	4,430	12,980

The hole in a hollow forged simple steel shaft of 15½ inches outside diameter is 7 inches. Nickel-steel hollow forged shafts having the same outside diameter may have a hole of 11¾ inches diameter. But for fear of any possible chance of buckling, the hole is made 9¾ inches in diameter. The propeller-shafts of the American Line steamers "St. Louis" and "St. Paul" are of nickel-steel; they will stand 42½ tons breaking-strain per square inch, and show 28 per cent. elongation and 50 per cent. reduction of area per square inch. The shaft of the "Iowa" will stand 45 tons breaking-strain per square inch, while 33½ tons is the limit in ordinary steel shafts.

"Here, then, is a material admirably suited to the shafting and engine-forging required by the marine engineer of modern high-service engines, and it is believed that as its merits become known its use will be widely extended. In the highest development of the modern marine engines, reduction of weight of all parts is of prime importance. This can only be accomplished by reducing sectional area. On the other hand, outside dimensions cannot be usually reduced without sacrificing necessary stiffness. We are, therefore, led to removing the metal along neutral axes, or, in other words, to the use of hollow forging. It is

evident that to farther reduce weight, as well as to increase the absolute strength of parts, the designer of marine engines needs a stronger material than that now employed; that is, a material having a greater elastic limit, but at the same time possessing such a degree of toughness as to insure resistance to sudden strain and shock. Simple steel strengthened and toughened by tempering and annealing will show, in specimens cut from the center of sections, say 3 inches to 6 inches thick, an elastic limit of about 45,000 pounds per square inch, an elongation of about 23 per cent., and a contraction of area of from 50 to 55 per cent. A farther and very pronounced improvement in strength and toughness can be obtained by the use of nickel-steel, tempered and annealed as above described. The use of nickel allows a reduction of carbon, makes the steel more sensitive to temper, and facilitates the tempering of irregular shapes. Specimens from nickel-steel forgings, tempered and annealed, will show uniformly an elastic limit of from 50,000 to 55,000 pounds per square inch, an elongation of 23 per cent. and above, in specimens 2 inches long by ½-inch diameter, and a contraction of area of from 55 to 60 per cent. In cases where, owing to thickness of sections and irregular shape, tempering is not advisable, nickel-steel will still show a higher combination of elasticity and toughness than any other material known, under the same conditions."*

ORDNANCE.

A complete set of nickel-steel forgings for an 8-inch gun has been made by the Bethlehem Iron Company for the Bureau of Ordnance, United States Navy, and is now being assembled at the Washington navy-yard. The average physical qualities obtained in these forgings in transverse specimens were:

	Tensile st. lbs. per sq. in.	Elastic limit lbs. per sq. in.	Elongation per cent.	Contraction of area per cent.
Tube, ...	93,200	58,300	21.2	42.0
Jacket, . . .	99,900	60,000	20.4	45.9
Hoops, . . .	109,100	68,200	20.5	46.9

Test specimens were 2 inches long by ½-inch diameter. Comparing with the average of qualities usually obtained in corresponding navy gun-forgings made of simple steel, the tensile strength shows an increase of about 10 per cent., with an increase in elastic limit from 22 to 28 per cent., while the contraction of area and elongation are but slightly reduced.

The Bureau of Ordnance found, while experimenting, that two small-arm barrels showed greater endurance than others. They were respectively of a very high-carbon steel and a steel containing about 4½ per cent. of nickel. The latter was fairly easy to machine, while the high-carbon steel was almost in-

* R. W. Davenport, Vice-President Bethlehem Iron Company, *Trans. Nav. and Marine Engrs.*, vol. i., 1893.

tractable. Consequently the Bureau decided to adopt nickel-steel for its small-arm barrels.* The great excellence attained by the Greener gun is attributed to the use of nickel-steel barrels containing 2.75 per cent. of nickel and 0.2 per cent. of carbon.

Other Uses.

It is evident that, besides the application to which nickel-steel is being put in armor-plate, gun-forgings and marine shafting, there is a still wider field open to its use for structural steel, heavy castings, car-couplers, car-wheels, boiler-plates, small pinions and knuckles, shear-knives, bicycle-spokes, gears for motors, and all varieties of work demanding hardness, toughness and malleability.

Plates of iron or steel and nickel, when laid together and heated to welding-temperature, may be rolled out into thin plates with a continuous nickel surface on both sides, or nickel on one side and iron or steel on the other. The union of the two metals is not merely a welding, but is of the nature of cementation, an actual alloy being formed to some depth below the surface of contact. There is a steam-vessel in New York harbor sheathed in part, as an experiment, with this material, fastened with iron nails. After eight months' constant service, the iron nails have corroded away, and all of the bottom, except the nickel sheathing, is corroded and foul, while the latter is as clean as when first put on. If nickel nails were used, it would seem as if nickel sheathing, or sheet-nickel, would make an ideal sheathing for all salt-water craft. This material is also used for lagging steam-cylinders, feed-water heaters, etc. It takes a beautiful polish, and is stronger than brass or copper.

The Niagara Falls Power Company has recently installed four 5000 horse-power electric generators coupled to turbine water-wheels. In this type of generator the periphery of the large rotating field travels at the rate of nearly two miles per minute. The bobbins are secured within a ring of nickel-steel that is forged without a weld, having an outside diameter of 139⅝ inches; inside diameter, 130 inches; width, 50¾ inches; weight, 28,840 pounds. This ring of nickel-steel is extremely light for its strength, and resists the centrifugal forces of this large field, while adding but little to its weight.

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* Private letter.

The Bureau of Steam Engineering, United States Navy, has decided to put nickel-steel boilers in the cruiser "Chicago," which is shortly to undergo repairs.

Nickel-Steel Wire.

Nickel-steel containing as much as 30 per cent. of nickel may be drawn into wire as easily as ordinary steel. Wire of this class, containing sufficient nickel to make the non-corroding qualities of the metal prominent, is especially adapted for hawsers and cable-service in salt water. A sample of nickel-steel wire, containing 27.8 per cent. nickel and 0.40 per cent. carbon, used as torpedo-defense netting by the United States Navy, gives the following physical test:

Diam. cross sec. Inch.	Area of cross sec. Sq. inch.	Reduced diameter. Inch.	Reduced area. Sq. inch.	Con. area. Per cent.	Elong. in 2 in. Per cent.	Load in pounds.	Breaking strain per sq. in. in pounds.
0.116	0.01057	0.106	0.0088	16.5	6.25	2100	198,700

The high tensile strength of this wire, with the comparatively small reduction in elongation and contraction of area, indicates extreme toughness; and at the same time it is not acted upon by salt water, so that it admirably answers the requirements of marine service.

Flange-Steel.

The Cleveland Rolling Mill made some flange-steel for the Canadian Copper Company, with and without nickel, for the purpose of making comparative tests of their physical qualities. The results are given in the table on the next page.

This nickel-steel shows an average increase of 11,400 pounds per square inch, or about 31 per cent. in elastic limit, and an average increase of 10,400 pounds per square inch, or about 20 per cent. in ultimate strength, without any perceptible effect upon the ductility, as evidenced by the percentage of elongation and contraction of area.

The Canadian Copper Company, at its works at Brooklyn, near Cleveland, Ohio, made a series of experiments on nickel-steel with varying percentages of nickel and carbon in an improvised acid-bottom open-hearth furnace. The heats amounted to about 1000 pounds of metal, made out of washed low-phosphorus pig and high-grade Bessemer ore. Nickel in metallic form was charged into the bath about one and one-half hours before

Comparative Tests of Nickel-Steel and Best Soft Flange-Steel.

(Specimens cut from plates.)

	Charge. Pounds.	Reduction of Area. Per cent.	Elongation in 8 inches. Per cent.	Elastic Limit. Pounds per square inch.	Ultimate Strength. Pounds per square inch.
I. Nickel-Steel. Containing C, 0.08; Mn, 0.36; P, 0.045; S, 0.038; Ni, 2.69.	Basic scrap, 9000. Low-phosphorus pig, 9000. 80-per cent. ferro, 165. 97-per cent. nickel, 540.	53.	23.25	....	64,080
		53.3	26.	47,100	66,370
		56.3	25.	44,700	66,000
		45.1	24.5	47,400	67,100
		54.4	26.	47,300	64,800
		49.7	23.75	48,900	66,200
II. Soft Steel. Containing C, 0.10; Mn, 0.27; P, 0.048; S, 0.039.	Basic scrap, 9000. Low-phosphorus pig, 9000. 80-per cent. ferro, 160.	45.6	26.	35,700	55,500
		45.8	26.	35,500	54,600
		52.9	27.5	32,800	53,900
		61.8	32.	34,060	52,500
		63.	27.	35,500	53,700
		63.	26.	37,900	56,500

tapping. Difficulty was experienced in controlling the heat, and other adverse conditions were encountered on account of the limited scale and lack of facilities in managing such a small furnace, which rendered it impossible to make steel of a uniform grade and show the degree to which a definite percentage of nickel in steel would be influenced by varying percentages of carbon, and *vice versa*. Still, the results of the physical tests of this steel may be of interest. The test-pieces were all taken from the center of the ingot, hammered to one and one-half inches square, and then turned down to a diameter of $\frac{5}{8}$-inch, with two inches between fillets, which were $\frac{7}{8}$-inch in diameter and threaded:

No. of Specimen.	Carbon. Per cent.	Nickel. Per cent.	Ultimate Strength. Pounds per square inch.	Reduction of Area. Per cent.	Elongation. Per cent.	Length. Inches.	Fracture.	Hardness in Lathe.
14	0.16	3.35	102,800	29.1	15.0	2	Silky.	Soft, UnA.
14	0.16	3.35	100,650	48.1	27.0	2	Silky.	Soft, A.
19	0.19	2.62	141,100	24.8	11.9	8	Gray.	Hard, Drawn.
13	0.22	2.05	88,880	34.6	20.5	2	Gray.	Easy, UnA.
13	0.22	2.05	84,650	55.4	31.5	2	Gray.	Easy, A.
13	0.22	2.05	83,040	58.2	25.1	8	Gray.	Easy, Drawn.
15	0.31	3.40	109,100	24.4	17.0	2	Gray.	Easy, UnA.
15	0.31	3.40	100,800	49.2	26.0	2	Gray.	Easy, A.
15	0.31	3.40	98,120	44.4	20.0	8	Silky.	Easy, Drawn.
41	0.51	4.93	127,075	27.10	16.0	2	Crystallized.	Hard, A.
24	0.54	3.20	131,200	12.7	10.5	2	Gray.	Hard, A.
24	0.54	3.20	134,400	36.7	14.3	8	Gray.	Hard, Drawn.
29	0.96	3.10	151,880	12.9	8.0	8	Gray.	Hard, Round.
34	0.91	3.10	138,000	22.3	9.88	8	Gray.	Hard, Round.

This steel was quiet in the moulds after tapping, set quickly without piping, and the ingots were smooth and clean. They were submitted to the same treatment in the hammer-shop and rod-mill as is given to ordinary steel. Through a mistake in getting numbers changed, the bars drawn through the rolls of the rod-mill received an extra annealing-heat. The conditions of the tests were as near alike as possible; the only exception being that the rods were pulled in the testing-machine* as they came from the $1\frac{1}{8}$-inch rolls, in 8-inch lengths, while the other test-specimens were 2 inches long and $\frac{5}{8}$-inch diameter.

The specifications of the Baltimore and Ohio R. R. for steel

* Otis Steel Company's Olsen machine.

tires, and the U. S. Navy Bureau of Steam-Engineering for crank- and propeller- shafts, connecting- and piston-rods and ordnance, are as follows:

Specifications of Baltimore and Ohio Railroad.

Grade.	Carbon, per cent.	Tensile strength, pounds per square inch.	Elongation in 4 inches, per cent.
I.	0.50 to 0.60	105,000	16
II.	0.60 to 0.70	115,000	14
III.	0.68 to 0.78	125,000	10

Grade I. is for passenger-engine tires, outside diameter, 60 inches; Grade II., for Consolidation, Mogul, etc., outside diameter, 45 to 60 inches; Grade III., for switching-engines, car-wheels, and all tires less than 46 inches in outside diameter.

A variation of 10,000 pounds in tensile strength above or below the above figures is permitted.

Specifications of the Bureau of Steam-Engineering, U. S. Navy.

	Tensile strength, pounds per square inch.	Elongation in 2 inches, per cent.	Contraction of area, per cent.
Propeller-shafts, ...	85,000	23	....
Crank-shafts, ...	58,000	28	....
Connecting-rods, ...	65,000	25	....
Piston-rods, ...	65,000	25	....
Ordnance, ...	85,000	18	35

It goes without saying, that, where other conditions are equal, soft or low-carbon steel possesses advantages over hard or high-carbon steel, as it is easier to machine, and (what is of greater importance) may be submitted to much rougher treatment, because it is not subject to the dangerous internal strains of hard steel. It is in this respect, especially, that nickel-steel, having the superior qualities of soft steel, fulfils the requirements of service sought for in hard steel, and offers to engineers the advantages of a material which will give greater strength with same weight, or equal strength with less weight, than any other at their disposal. Comparing the accepted standard of mild steel with nickel-steel having approximately the same carbon-contents, we have:

	Tensile strength, pounds per square inch.	Elongation, per cent.	Contraction of area, per cent.
Ordinary steel, ...	65,000	23 in 8 in.	48.0
No. 13 nickel-steel (2.05 pr cent. nickel), ...	84,650	31.5 in 2 in.	55.4
No. 14 nickel-steel (3.35 pr cent. nickel), ...	100,650	27.0 in 2 in.	48.1

We have here nickel-steel, containing less than 0.2 per cent. carbon, and 3.35 per cent. of nickel (annealed), that more than meets the specifications of the Navy Department for ordnance, shafting, etc., and of Grade I. for steel tires on the Baltimore and Ohio R. R.

For Grade III., requiring high-grade steel, we make the following comparison with nickel-steel, annealed, containing 0.20 per cent. less carbon than the required carbon in plain steel:

	Tensile strength, pounds per square inch.	Elongation, per cent.
B. & O. R. R. steel tires, Grade III., .	. 125,000	10 in 4 in.
No. 24 nickel-steel, ...	. 134,000	14 in 2 in.

By 2.0 per cent. of nickel (No. 13) the tensile strength of mild steel is raised 30 per cent., and by 3.35 nickel (No. 14) 41 per cent., without any appreciable change of elongation or reduction of area. "The presence of 4.7 per cent. of nickel increases the tensile strength 35 per cent., and the elastic limit 75 per cent., while the elongation and contraction of area is practically the same."*

In reviewing the results of these experiments, corroborated by the experience of others, it is found that better results are obtained by using more rather than less than 3 per cent. of nickel. The tensile strength and elastic limit of steel increases with the percentage of nickel, up to the point of extreme hardness in machining, and the percentage of carbon has everything to do in raising or lowering this property of nickel-steel, as much as in ordinary steel.

Torsion-tests of these specimens were made by the Standard Tool Co., Cleveland, Ohio, as follows:

No. of specimen.	Carbon, per cent.	Nickel, per cent.	Torsion breaking-point, in pounds per square inch.	Degrees of twist in 3 inches before breaking. 360 degrees = 1 full twist.	
14,	. 0.16	3.35	2325	360	....
19,	. 0.19	2.62	2150	130	Split.
13,	. 0.22	2.05	2434	240	Twisted off.
15,	. 0.31	3.40	1807	355	....
41,	. 0.51	4.93	2200	120	....
24,	. 0.54	3.00	1200	60	Split.
29,	. 0.96	3.10	1700	60	Split.

* Riley's Experiments.

The specimens in these torsion-tests were 1¼ inches square. A number of the specimens were found to be checked and laminated in structure.

In a cold-bending test of a specimen 2½ by 2½ inches (full thickness of wall of forging), 18 inches long, under hydraulic press through 180°, the ends met within ½ inch; the greatest distance between sides was ⅞ inch. There was only one slight crack, in one corner on the inside of the bend.*

The percentage of nickel in all the government work herein referred to is 3.25 per cent., with carbon at about 0.2 per cent. It is not improbable that familiarity with working and cheapening the cost in manufacture will permit the percentage of nickel to be considerably increased above this figure to good advantage. It has been the practice in this country to charge the nickel into the furnace in the form of nickel oxide enclosed in sheet-iron boxes. In other countries, pig- or ferro-nickel is used. Some steel-plants use metallic nickel, which offers this advantage over the oxide, that less nickel slags off. The best results are obtained in the basic open-hearth furnace. Several of the Pittsburgh steel-works use nickel as an alloy for steel, but are not yet prepared to make a special feature of nickel-steel castings outside of government work. The Bethlehem Iron Company, having enlarged its plant, has special facilities for making nickel-steel in any desired form or size for the general trade, besides taking large government contracts.

It is obvious from the foregoing data, which briefly summarize the present status of the metallurgy of nickel, that the field for the use of nickel is one of magnitude, and that the era of its development has only just commenced.

The results herein given are accompanied with authorities, so that they may be followed more in detail by those desiring to study the subject further and to discuss the statements offered in this paper.

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Cinnabar in Texas.

BY WILLIAM P. BLAKE, MILL ROCK, NEW HAVEN, CONN.

(Florida Meeting, March, 1895.)

THE literature of the occurrence of quicksilver-ore in the United States does not contain, so far as the writer is aware, any mention of the locality herein described.

In the preliminary report * upon the resources of the trans-Pecos region of Texas, von Streerwitz gives a long list of minerals observed as occurring there, but cinnabar is not mentioned. In the second report upon the same region † mention is made of the reported existence of cinnabarite in one of the mountain ranges north of the Sierra Carrizo and the Bofecillos, but the author adds: "In spite of my careful examination of the float, I have not yet found any traces of this metal (quicksilver) up to the present time." Prof. Dumble, also, in

* *Geol. Surv. of Texas*, E. T. Dumble, 1889, p. 225 (first report).

† *Geol. Surv. of Texas*, E. T. Dumble, 1890, p. 713 (second report).

his general report,* says of mercury, "that, like tin, this metal has been reported from several localities, but up to the present we have not succeeded in verifying any of the reports or of finding any traces of it."

The later reports of the Texas survey contain no further reference to this subject, nor is any occurrence in Texas noted by Becker in his exhaustive monograph upon the quicksilver-deposits of the United States.

Early in the year 1894, Mr. George W. Wanless, of Jimenez, Mexico, agent at that point of the Rio Grande Smelting Works, having learned that some Mexicans had obtained very rich cinnabar in the mountains of Texas, a few miles north of the Rio Grande, undertook, together with Mr. Charles Allen, of Socorro, N. M., an exploration of the region, with the result of finding the cinnabar-deposits and locating them for development. My attention was directed to them through Mr. James P. Chase, of Socorro, with whom I visited the locality in the month of August last. About the same time a notice of the discovery was printed in Los Angeles, Cal.,† and it was also mentioned in one or more of the papers in El Paso, and later in the *Manufacturers' Record*, published in Baltimore.

The locality is in the southern portion of the part of Texas within the Big Bend of the Rio Grande river, about 80 or 90 miles south of Alpine station, and 90 or 100 from Marfa station on the Southern Pacific Railway. It is 50 or 60 miles from Presidio del Norte, and about 10 or 12 miles from the Rio Grande. These distances, it will be noted, are approximately stated, as there has not been any survey of the region. The longitude is about 104° W. and lat. 29° 30′ N. The cinnabar is best reached from Marfa by team through an open country, with a gradual descent from the Marfa table-land to the Rio Grande valley, following first the valley of the Alamitos and then over a low divide to the Tres Lenguas, which is followed southwards, generally between the flat-topped hills of the mesas on each side, until nearing the Rio Grande, where the road winds among higher and more rugged hills. The last six miles of the route is impassable for wagons, and the cinnabar camp is reached by a pack-trail which turns westwards from the

* *Geol. Surv. of Texas*, E. T. Dumble, 1890, p. lxix. (second report).

† *The Bullion*, Aug. 14, 1894, vol. xii., No. 16, p. 3.

wagon-road and leads across a country much broken and intersected by dry "washes" or creek beds.

The hills are low, but are much broken by escarpments of nearly horizontal strata of Cretaceous limestone. The elevation of the camp is shown by the aneroid barometer to be 3250 feet above tide. The valley of the Rio Grande, marked by its fringing groves of cottonwood-trees, is in full view for several miles. The mountains across the border in Mexico are also clearly seen, as well as the high range on the southeast known as Los Chisos, culminating in Emory's Peak, and the peculiarly-shaped peaks known as the "Mules' Ears"—all noted land-marks. Major Emory, describing this region in a few words, says:

"The Rio Bravo, accommodating itself to the geological formation of the country, makes, between the 100th and 104th meridian of longitude, two great bends nearly symmetrical, one to the south and the other to the north. The area included in the southern bend is one vast Cretaceous bed, upheaved by igneous protrusions, sometimes forming ranges of mountains, as the Limpia range, and at others isolated peaks, like Gomez Peak and San Jacinto."*

From Marfa to the Tres Lenguas the direction is nearly southeast, following a widely eroded valley in table-lands, which are generally capped with a hard layer of basaltic lava, seen to the best advantage at the Alamitos rancho and in the Church Mountains, near Collinson's rancho. An isolated conical mountain, rising from the broadly eroded country to the eastward of the Church Mountains, has a flat top and is evidently a remnant of the former mesa, left standing as a monument, as if to show what an enormous amount of material has been swept away to the Gulf by erosion and degradation. The edges of the horizontal beds can be seen from a distance, and the mountain, known as San Diego Peak, doubtless affords a very complete and interesting section of the entire series of beds from the lava-cap to the lower strata of the Cretaceous.

The beds of which this peak and the mesas along the Alamitos and the upper branches of the Tres Lenguas are chiefly formed are remarkable for their whiteness and homogeneity, and appear to consist chiefly of an indurated volcanic mud. It is an amorphous mass, in which there is a large amount of clay and silica; but it is without well-defined structure of strati-

* *Rept. U. S. and Mex. Bound. Sur.*, i., 88.

fication. It is remarkable for the general absence of oxide of iron. It is fusible and would appear to be a mass resulting from the breaking up of feldspathic rocks. The thickness is probably not less than 500 feet; and it extends over a wide area, east and west, as far as the edges of the high mesas can be seen. The general uniformity of composition of this deposit is broken towards the top of the mesas by a bed of conglomerate and breccia, 10 feet or more in thickness, made up chiefly of red and brown porphyritic rocks. The masses being, in part, well-rounded, show the action of currents of considerable force and extent. This stratum contrasts strongly with the white sediments above and below, and makes a dark-colored belt or band through the hills visible for miles on either side.

Of the geological age of this series of beds under the lava, it is impossible from the limited observations and the entire absence of fossils to write positively;* but it is my opinion that the Pliocene and Miocene Tertiary are represented, and that these volcanic deposits should be correlated in time with those of the upper Gila, of Central Arizona, and those of the gold-region of the Sierra Nevada in California.

In descending the valley of the Tres Lenguas there is a marked transition from the volcanic beds to those of unquestionable Cretaceous age. At first thick masses of finely-bedded blue and yellow shales are encountered, and in the broad flat surfaces countless casts of *Inoceramus* reveal their proper horizon. The strata, at first lying apparently horizontal, are found to be cleft in various directions by faulting-planes, with large blocks partly up-turned and evidences of extreme lateral pressure, by which the shales along the faults are buckled upwards and crushed.

The shales are succeeded by limestones, massive, light-colored, nodular, and rugose in structure, and filled with large specimens of *Exogyra* and other characteristic Cretaceous fossils.

As the cinnabar-locality is approached, the Cretaceous strata are much more broken and uplifted in monoclines dipping west. Older and crystalline rocks appear and there are intrusive dikes. Compact blue limestones are found, with an abund-

† Upon this subject the reports of Prof. Dumble of the Texas Survey may be consulted.

ance of *Gryphea* and some ammonites. There is a large development of a yellowish-brown limestone, largely made up of the foraminifera *Nodosaria texana* (Conrad), marking a well-known geological horizon in eastern Texas. According to Prof. Dumble, of the Texas Survey,* its position is the upper portion of the Arietina clays, though in the trans-Pecos region it seems to occur near the base of the Washita division. This is the nearest well-defined fossil horizon I have found contiguous to the cinnabar deposits with the exception of what might be inferred from a fossil *Pecten* discovered in one of the cinnabar-openings.

The cinnabar occurs both in massive limestone and in a siliceous shale and a white earthy clay-like rock, and in part in a true breccia of grayish-white siliceous shale, dense and compact, imbedded and cemented in a red and chocolate-colored ferruginous mass, also dense and hard. The white blocks, or included fragments of the shale, exhibit a concentric arrangement of coloring by oxide of iron disposed in bands and thin sheets, deposited in the substance of the shale by the absorption of ferruginous solutions, penetrating from without inwards along the surfaces of the fragments. These deposited coatings or layers conform in general to the exterior forms of the masses, and succeed each other like the concentric layers seen in agates and chalcedony. The colored depositions may also be seen surrounding tube-like or thread-like channels, which have permitted the inflow of solutions bearing not only iron salts, but also those of quicksilver, and leaving behind, in the substance of the rock, layers of iron oxide and of cinnabar concentrically disposed.

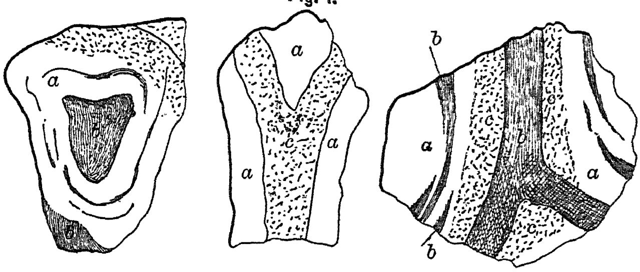
Fig. 1, engraved from fragments of the homogeneous white rock, shows the general type of distribution of the bright crystalline grains of cinnabar in association with vein-like and concentric layers of iron-oxide.

While the genesis of the cinnabar is here shown to be essentially like that of the iron oxide, it is smaller in quantity, and is so far separated from the ferruginous bands as to show a great difference in the conditions of deposition. The cinnabar is more generally crystalline than amorphous; it is not found in such continuous coatings or layers in the white shale as the iron oxide, but is in distinctly separate grains and small but

* *Geol. Surv. of Texas*, 2d Annual Report, pp. 719–736.

brilliant rhomboidal crystals, having the brilliant red color characteristic of vermilion. There are also considerable masses of snow-white argillaceous rock, seemingly the result of alteration of shale by infiltration, in which cinnabar is found in minute crystalline grains spread in bunches here and there through the mass, and often not observable until the mass is rubbed or bruised with a pick or hammer; when the red color of vermilion appears. In such masses there is apparently a complete absence of ferruginous matter. The soft, white, chalk-like masses of rock do not appear to be so favorable to the crystallization of the cinnabar as the more dense and siliceous portions of rock, resembling chert or flint, where the cinnabar is

Fig. 1.



Texas Cinnabar-ore: *a*, white rock; *b*, iron oxide; *c*, disseminated crystalline cinnabar.

in distinctly-formed crystals sprinkled through the rock, much like the occurrence of cinnabar in the siliceous gangue of the deposits of Buckeye Rancho, California.

In addition to these disseminated crystalline granules in the brecciated shale and in the more massive white rock, there are amorphous bunches of cinnabar found in the shales and in the limestones and the breccia. This cinnabar is not, however, in hard masses like those of the New Almaden mine in California, nor is it in veinlets, as there found, traversing the rocks; but it is soft and friable and has a light vermilion color. Calcite is associated with it, but, so far as observed, no petroleum or bituminous exudations. Some of the larger masses of cinnabar bunches, weighing two or three pounds of nearly pure mineral, were taken out of an open cut, where the shale appears to be the parent-rock; but I noted also some small bunches of the cinnabar in the compact blue limestone.

There are several points on the line of about 1000 feet in length, along a shallow ravine, where open cuts have been made to a depth of a foot or two, revealing the presence of cinnabar in each and in the soil mixed with the croppings. This linear distribution of the cinnabar is indicative of a vein-like occurrence, or it may be the result of a cropping of a certain bed or stratum. The openings which had been made were not deep enough to show conclusively the real conditions of occurrence. In some places the appearances favored the conclusion that the ore is interstratified or bedded; in others it seems to occur along a fissure or fault-plane, and it is most probable that both of these forms of occurrence will be found to exist.

There is a second line of cropping of cinnabar a few rods north of the first and higher up the hill, and in the midst of the hard limestones. This is in the midst of a well-defined breccia of iron-oxide, and the masses of cinnabar are closely associated with it. Masses of iron-oxide rock and cinnabar weighing a hundred weight or more can be broken out from croppings here; but no work has been done to develop this ground in depth. The cinnabar-croppings may be traced for a few feet each way, and the breadth does not exceed 18 inches or 2 feet. This occurrence does not appear to be connected with the series of croppings before described, and it has a more decided resemblance to a fissure deposit or impregnation.

The existence of several outcrops of a ferruginous breccia, with and without cinnabar, is indicative of breaks in the beds in the nature of fissures (fault-planes, probably), accompanied by rupturing and crushing of the rocks by violent movements under pressure. Other evidences of fissuring and of metalliferous impregnation through the fissures are visible in the neighborhood in the many vertical cracks in the limestone strata marked by the lateral deposition of oxide of iron on both sides. Such deposits are extensive and show that there has been an abundant supply of iron-bearing solutions. Whether these solutions flowed from below upwards or from above downwards it is not essential to determine; but the source of such solutions is indicated in one place, not far distant, by very considerable deposits of iron-pyrites in a blue clay or bluish-green shale cropping out a short distance south of the cinnabar-

deposits. This pyrite is compact, granular, non-crystalline and bronze-like, and is apparently in nodular masses in the shale. The relations of these deposits of pyrite to the cinnabar were not further ascertained and can only be theoretically assumed. We may in this way suppose that the cinnabar is present in the pyrites, and that it is carried in solution and deposited, together with the iron-oxide, in the rocks, where the conditions are most favorable. Careful chemical investigations can alone determine whether the pyrite contains quicksilver, and the nature of the chemical changes of decomposition and recomposition resulting in the formation of the cinnabar.

The phenomena all point to a formation from aqueous solution rather than to a deposition from vapor.

Leaving theories of origin aside, the practical question to the miner and metallurgist is as to the best place in which to sink for the better development of the ore in quantity. This place would appear to be at the brecciated cropping on the hill in the limestone. Whatever the origin of the ore may be, this place seems the most promising; as limestone, being the more soluble rock, may contain large bunches below, where the ore has been accumulated by replacement. On the other hand, we have seen that the insoluble porous shale-rock offered favorable conditions for the deposition of the cinnabar without the exhibition of the phenomena of replacement, the conditions here favoring the gradual concentration of mercuric solutions by evaporation from the pores of an insoluble containing-rock.

In considering the source and origin of the cinnabar, we should not lose sight of the fact that there is an intrusion of doleritic rock near by, and that this probably has direct and close connection, not only with the disturbance of the strata, but also with the source of the metalliferous impregnations.

The conditions for working these deposits of cinnabar are not as favorable as could be wished. The brilliant color of the ore would permit of its being utilized as vermilion; but there is no water near the place for concentrating it. Considerable ore could be taken out of the loose earth along the main croppings if water could be had to wash with. A supply of wood for fuel can be had along the Rio Grande, and could be delivered at the mines for probably $5 or $6 per cord. Although a considerable quantity of high-grade ore which

would bear transportation could be selected by culling, there would remain a larger quantity of low-grade which would be practically useless. There has not yet been sufficient work done on the croppings to show satisfactorily what quantity of ore of a desirable average percentage can be expected.

The Northeastern Bituminous Coal-Measures of the Appalachian System.

BY GEORGE S. RAMSAY, MCKEESPORT, PA.

(Florida Meeting, March, 1895.)

THE Appalachian system contains the largest area of all known Carboniferous coal-fields. Beginning near the north line dividing Pennsylvania and New York, it extends southwest through West Virginia, southeastern Ohio, Kentucky and Tennessee and terminates in Alabama—a distance of about 950 miles. The northeastern part of this field will be considered in this paper. The general dips of the measures are toward a trough about parallel to the Ohio river, from the Allegheny mountains and ridges on the west. The Potomac field is an exception; the dip in this case being northeast, the folding to the east of the Allegheny mountains in Pennsylvania being of a greater degree and diminishing southwestward.

The cause of the general dip to the west of the mountains seems to be the westward movement of the thrust east of the mountains, with the large Nittany fold including other folds to the east of the escarpment, raising the mountains high above the coal-fields to the west, making a general slope toward the Ohio river southwest, and crossing waves of anticlinals westward. The synclinals contain the coals, while in many cases on the top of the folds or anticlinals some of the veins (or, in some cases, all) are eroded. The Coal-Measures of this field are generally divided into three principal divisions, known as the Upper, the Barren, and the Lower Coal-Measures, but there is a tendency to divide them into five groups, which is more convenient. Beginning at the bottom, the first would be the Conglomerate; second, the Lower Coal-Measures; third, the Barren; fourth, the Upper Coal-Measures, and fifth, the Upper Barren or Permian Measures.

1. *Conglomerate.*—The Pottsville conglomerate was formerly considered the base of the Coal-Measures containing no workable coals. But in recent years this theory has been exploded. Later investigations have developed the fact that some valuable coal-beds have been and are now worked on the outcrop in different parts of the Appalachian coal-field in the conglomerate. The thickness of the measures varies in different localities. In Ashburner's report (R R, page 186), we find them to be in Elk county, Pennsylvania, 170 feet in thickness, containing a bed of coal 4 feet thick, 35 feet from the top of this group. The same measures on the Blackwater, a tributary of the Cheat river in West Virginia, are 650 feet thick and contain several small veins of coal; the upper one being 2 feet 7 inches thick and about 300 feet from the top of the conglomerate or base of the Lower Coal-Measures. Other small veins are found near the bottom of this division. Prof. White reports these measures 1400 feet thick at Nuttalburg, West Virginia, indicating the thickening of the measures going southwest along the outcrop. He also reports the same measures 1400 feet thick on Crane creek, Mercer county, West Virginia; and at this point the veins of coal have thickened considerably.

At Pocahontas we find the veins have also thickened; but there is some doubt as to the identification of the Pocahontas bed with that of Quinnimont at New River. It will be plainly seen that the veins have increased in thickness with the thickening going southwest along the outcrop to the Virginia line.

In the northwestern part of the Appalachian field, we find the Mercer group. It belongs to the same measures, but is only about 300 feet in thickness and contains workable beds of coal, the Sharon being the principal bed and about 4 feet in thickness. In the center of the field, taking Westmoreland, Armstrong, and Washington counties in Pennsylvania, we find the same measures 225 to 300 feet in thickness and carrying no veins of coal. From explorations in different parts of the field, the only workable beds that show persistency seem to be around the outcrop or border of these measures.

As far as we have knowledge of the center at present, it contains no workable coal, although in some localities, traces are found in places. It is not difficult to identify the Pottsville conglomerate in traveling through Pennsylvania or West Virginia. When these rocks come to the surface, wild and rugged

scenery is found with rapid rivers and high waterfalls, as in the cases of the west branch of the Susquehanna river, the Moshanon, near Karthaus, and the Youghiogheny river above Connellsville in Pennsylvania. The Cheat, Blackwater and Kanawha rivers in West Virginia, furnish both rugged scenery, waterfalls and rapids. We generally find the conglomerate impediments in the rivers after getting below the lower Coal-Measures. This series of rocks is composed generally of very hard sandstones, the grains of which are well cemented together. The structure is made up of sand and pebbles, the latter varying in size from that of a pea to that of a large marble, and larger, increasing and decreasing in different localities. Practically these rocks are indestructible and furnish the best building material for large stone structures. This group is a good key for the location of the different divisions of the entire column of Coal-Measures, when exposed in a mountainous country, or when the diamond drill is used, where the lower measures are covered by the upper.

Lower Coal-Measures.—These lie on the Pottsville conglomerate and below the Mahoning sandstone and contain valuable beds of coal. At present they are principally worked on the outcrop, or when the Pittsburgh bed is not present. The thickness of the beds in this series varies in different parts of the field, and from later investigations by the drill, it seems they are thicker on the outcrop, than in the center of the field. The veins vary in thickness in different localities, from a few inches to 5 and 6 feet, and are known by the following names, beginning at the bottom, next to the conglomerate: Clarion, Lower Kittanning, Upper Kittanning, Lower Freeport, Upper Freeport. Frequently local veins make their appearance; but the above-named beds are the principal ones. The entire series varies in different parts of the field; but the tendency is to greater thickness going south. In northeastern Pennsylvania it is about 150 feet. Coming south, we find at Karthaus, Clearfield county, an increase to about 250 feet; and the group is still thicker at Salisbury, Somerset county, where it measures 300 feet.

On the Blackwater, a tributary of the Cheat river, West Virginia, we find it 320 feet; and in southwest Virginia, 1000 feet in thickness. The thickness of this series in the center of the field, going southwest, varies in about the same proportion

as the eastern outcrop. In these measures are locked up, for future generations, valuable beds of coal, right under the Connellsville coke-plants, and the surrounding country, and it will be only a matter of time when the lower veins will be mined. The drill has disclosed, and will disclose, the workable beds of coal as they will be required. These veins will be found from the mountains near Connellsville down the Youghiogheny river to McKeesport; from the source of the Monongahela to Pittsburgh; following the Ohio river to where the outcrop of the Appalachian coal-field is crossed by that river; and from Pittsburgh north to where the Appalachian crops out on the Allegheny river. The same measures and beds of coal can also be found east of Pittsburgh, under the Pennsylvania railroad, to where they crop out in the Conemaugh river, where the lower beds are worked quite extensively, from Johnstown to Galitzen. In all probability, the Freeports can be reached in the neighborhood of Pittsburgh and Connellsville at a depth of 550 feet below the Pittsburgh bed. The Kittanning and Clarion beds are within a depth of 800 feet. As a rule, the veins are not all workable in the same locality.

Frequently, as one of these beds increases in thickness, some of the others decrease. In the Clearfield region, the Upper Freeport shows a thickness of 2½ feet, and the Lower Freeport from 3 to 5 feet, the Lower Freeport being the great Moshanon vein. From this vein the greater part of the coal taken from Clearfield county has been mined.

The next in the column are the Upper Kittanning (3 feet); the Lower Kittanning (4 feet); and the lowest, the Clarion (3 feet 6 inches). From a section in Somerset county we find the Upper Freeport to be 14 feet, with a shale 8 feet thick in the center; the Lower Freeport being 3 feet, the two Kittannings in this section being absent, and the Clarion reaching a thickness of 6 feet. In Westmoreland county, a section from Laurel run, Ligonier township, shows Upper Freeport 4 feet; Lower Freeport, 6 inches; Upper Kittanning 6 feet 7 inches, with a band of sandstone in the center; Lower Kittanning 5 feet 2 inches (very slaty); Clarion beds, 2 feet thick. In Beaver run, in the same county, a section shows: Upper Freeport, 3 feet 6 inches; Lower Freeport, 6 feet; Upper Kittanning, 6 feet; Lower Kittanning, 4 feet; and the Clarion bed replaced by shale. At the head-

waters of the Blackwater, West Virginia, we find the Upper Freeport 8 feet, with a piece of bone in the center; Lower Freeport, 18 inches; Upper Kittanning, 1 inch; Lower Kittanning, 11 feet (the lower part of the bed containing a piece of poor coal and fire-clay, 4 feet 6 inches); Clarion, 2 feet 6 inches.

Barren Measures.—These are third in the ascending column, beginning at the bottom of the Mahoning sandstone and terminating at the base of the Pittsburgh coal-bed. As the name "Barren" indicates, this series contains very little workable coal. The beds found in this group are, namely, the Mahoning, Masontown, Bakerstown, Elk Lick, and Little Pittsburgh. The place of the Mahoning coal is in the center of the Mahoning sandstone. When this vein is present, the Mahoning sandstone is thereby divided into the upper and lower. The vein is generally about 35 to 40 feet from the base; its thickness is from a streak to 2 feet. In some localities shale will be found in place of the vein. The Masontown coal, second in the group, is about 110 to 120 feet from the bottom of the Mahoning sandstone, and the Bakerstown about 300 feet, and the Elk Lick about 390 feet above the same horizon. These coals run from 1 to 2 feet in thickness. The above beds, as a rule, are high in ash, although there are some exceptions. The Little Pittsburgh vein is located from 65 to 70 feet below the Pittsburgh bed, and near the top of the Barrens. This bed, like the others of the group, is generally of poor quality, and from 18 inches to 2 feet in thickness. It is exposed in a creek north of Larimer station, on the Pennsylvania railroad, and at Fairfax Knob, West Virginia. At these two places the coal is of good quality. This vein seems to be persistent in the northeast part of the field, when the measures are present. The entire thickness of the Barrens averages about 600 feet.

The Mahoning sandstone, the base of the Barren Measures, is a hard and indestructible bed, averaging 100 feet in thickness. The hardness of this stone has been a great protection to the lower Coal-Measures, resisting moisture and protecting the softer measures below. It generally caps the top of the Allegheny mountains coal-fields.

Upper Coal-Measures.—This fourth number from the base of the conglomerate series begins at the floor of the Pittsburgh coal-bed, and terminates at the top of the Waynesburg vein.

The following are the principal coal-beds in this series: the Pittsburgh vein, at the bottom; the Redstone coal, about 50 feet above the Pittsburgh floor; the Sewickley bed, nearly 130 feet; the Uniontown vein, approximately 270 feet; and the Waynesburg coal, which is the uppermost one in this division, making the Upper Coal-Measures about 270 feet thick in Fayette and Westmoreland counties, in Pennsylvania. These measures, like the lower measures, thicken toward the south into West Virginia, to about 400 feet. At the northwestern outcrop they are only 200 feet thick. The most important bed in this group is the Pittsburgh bed. Originally this bed covered a great area, extending east from the western outcrop in Ohio, to beyond the Allegheny mountains. Only about 20 per cent. of this great vein now remains in Pennsylvania. We find east of the Youghiogheny river only strips or patches, confined to eight counties in Pennsylvania, namely, Greene, Washington, Fayette, Westmoreland, Indiana, Armstrong, and Somerset counties, with a little knob east of the Allegheny mountains, at Broad Top. We also find the same vein and measures in Maryland and eastern West Virginia, known in those two States as the Big Vein, and mined extensively near Cumberland.

South of the Pennsylvania line, at the headwaters of the Monongahela river, in West Virginia, a large territory of the Pittsburgh bed, extending south through that State, is practically undeveloped. Along the Monongahela river, south of Pittsburgh, the vein is high in the hills; further south, and up the river, in some places, it passes under the river bed in the synclinals; but generally the river has cut through the bed, giving excellent opportunities for mining without expensive shaft sinking and pumping. We also find that this bed has been eroded away from the ridges or divides, leaving only strips and patches on hill-tops. These patches are to be found in Ligonier valley, Westmoreland county, Myersdale, Somerset county, and at the northeastern end of the strip, running from Latrobe to Blairsville intersection on the Indiana side of the Conemaugh river; and there are also a large number of patches at the northern outcrop of Allegheny county, Pennsylvania.

The largest strip outside of the main body is the Connellsville coking-coal field, beginning about at the West Virginia line near Smithfield and Georges Creek. This coal-basin ex-

tends through Uniontown, Mount Pleasant, and thence into the Latrobe district, and terminates near Blairsville Junction, giving a length of about 60 miles and a width from 3 to 4 miles. To the west of this basin is the Blairsville anticlinal, crossing the Loyalhanna at Brady's old mill-stand and the Pennsylvania railroad a short distance east of Carr's Tunnel. The arch is high enough to expose the Barren Measures. West of this anticlinal is the Greensburg coal-basin, containing the Pittsburgh bed, and being about 4 miles wide and 13 miles in length, with a small patch to the north near New Alexandria. West of the Greensburg basin is the Saltsburg anticlinal, running across Westmoreland county and terminating near the mouth of Jacob's creek, where it flattens out; and west of the Saltsburg anticlinal is the Pittsburgh gas-coal, with one fork of the Pittsburgh bed running through, near Port Royal and Manor, to a point on the Kiskiminetas river below Saltsburg. West of this, and joining it to the main body at the southwest end of the limb, and from this point south and west, is the main body of the Pittsburgh Coal and Upper Measures, extending west into Ohio and south into West Virginia.

From the time of the westward movement, throwing up the Allegheny mountains and forming ridges and arches, this bed has been for thousands of years slowly disappearing by erosion, as the isolated remains in different sections of the Appalachian field indicate. The bed varies in different localities, but is quite easily recognized, as the following sections will indicate.

	I.		II.		III.		IV.	
	Ft.	In.	Ft.	In.	Ft.	In.	Ft.	In.
Roof coal, ...	4	10	3	8	*a* 3	4	5	8
Main clay, ...	0	10	0	5	*b* 4		*b* 1	0
Main coal, ...	5	10	8	0	10		13	7
Total, ...	11	6	12	1	17	4	20	3

a. At one point. *b.* Slate.

I., Near Elizabeth, Pa.; II., At Uniontown, near Connellsville; III., Near Salisbury, Pa.; IV., near Westernport, Md.

This shows the tendency of the vein to thicken east of the Ohio river to the eastern outcrop of the Appalachian coal-field.

From 40 to 50 feet above the Pittsburgh coal there is a vein of coal known as the Redstone bed. In Westmoreland and Fayette counties it is from 3 to 4 feet thick. It is also reported

in the Salisbury basin as being from 1 to 4 feet, and in the vicinity of Piedmont as over 4 feet 6 inches thick. Where the Pittsburgh vein has been mined and the pillars drawn, this bed of coal has been so far destroyed as to have no practical value.

A second vein in this series, about 125 feet above the Pittsburgh bed, is the Sewickley coal, which varies in thickness in different sections of the country. In Fayette county it frequently measures 3½ to 5 feet, and thins out northward. In the Salisbury district it is only about 2 feet thick, but in the Cumberland region it increases to 5 feet of good coal.

The third vein in the column—about 275 feet above the Pittsburgh bed, and known as the Uniontown coal—varies from the thickness of a knife-blade to 3 feet in different parts of the field. In the upper measures of the Salisbury basin it shows a thickness of 3 feet, but is slaty.

The fourth in this column is the Waynesburg coal, which is the top bed in this series. This bed is of very little commercial value, since it generally contains large quantities of sulphur and is very slaty. It is chiefly used for domestic purposes. It varies from 1 to 3 feet in thickness. In Greene county, Pennsylvania, it thickens to 8 feet.

Upper Barren Measures.—These begin at the roof of the Waynesburg coal, and extend up to the surface. Prof. White gives the highest point of these measures at 1100 feet, and the highest section of rocks at the headwaters of Dunkard Creek, a stream near the West Virginia and Pennsylvania line. In Greene and Washington counties a large area of these measures is found, with isolated patches in Fayette and Westmoreland, in Pennsylvania. A large field lies south of the Pennsylvania line, containing these measures. The main coal-bed in this series is the Washington, about 175 feet from the top of the Waynesburg coal. It varies from 15 inches to 10 feet. In Westmoreland and Fayette this vein presents about 5 feet of thickness, but the bed is slaty. In South Huntingdon township, Westmoreland, the bed shows a thickness of 9 feet, but is in poor condition, being composed of alternating layers of slate and coal. Other small streaks of coal make their appearance in these measures, but, as yet, very little coal of commercial value has been found in them; hence the reason for the name "Upper Barren Measures."

Note on a Proposed Scheme for the Study of the Physics of Cast-Iron.

BY WILLIAM R. WEBSTER, PHILADELPHIA, PA.

(Florida Meeting, March, 1895.)

In view of the great interest now taken in the tests of cast-iron and details of foundry practice, with the number of investigators at work, and recent improvements in the methods of research, it would seem that the time is ripe to attempt the solution of some of the many problems with which iron founders have to contend. Recent papers and discussions on these subjects* have opened up a large field for investigation, and have emphasized the importance of many considerations generally overlooked. With the view of promoting a comprehensive and systematic discussion, I have attempted to tabulate, in convenient form, some of the most important points for investigation, on the general plan suggested, with such good results, by our former President, H. M. Howe, for the discussion on the Physics of Steel, at the Virginia Beach meeting, last year. This table can, no doubt, be modified or enlarged to advantage, and put in shape to serve as a guide or reminder to all who are interested in this line of work.

Suggested Lines for Discussion and Investigation.

I.—Correspondence between chemical composition and melting point, fluidity, shrinkage, fracture, chill, micro-structure, and other physical properties.

II.—*Influence of:*

1. Cupola mixture, use of steel and other scrap, oxidized or clean material,
2. Manner of melting, flux, etc.,
3. Casting temperature,
4. Manner of handling melted metal and method of casting,
5. Size and form of casting,
6. Kind of mould, green sand (under different conditions of ramming, amount of moisture, and skin-dried), dry sand, loam, and chills,
7. Rate and mode of cooling castings,
8. Manner and temperature of heating for annealing,
9. Additions of nickel or aluminum,

on

A. Fracture.
B. Micro-structure.
C. Physical properties.
D. Shrinkage.
E. Chill.
F. Residual stress.
G. Condition and quantity of carbon and other elements.

* Among other references in the *Transactions* to this subject, I would mention besides the papers and discussions on the physics of steel, and on methods of analysis, the papers of Mr. John B. Pearse (iv., 157); Sir Lowthian Bell (v., 77); Edward Gridley (xii., 91); F. P. Dewey (xvii.,460), and W. J. Keep (xvii., 253, 683; xviii., 102, 458; xx., 291; and xxiii., 382).

III.—*Segregation as affected by:*
1. Composition.
2. Casting temperature.
3. Rate of cooling.
4. Size and shape of casting.

IV.—*Blow-holes, their volume and position as affected by:*
1. Composition.
2. Casting temperature.
3. Casting pressure.
4. Rate of cooling.
5. Size and shape of casting.
6. Special additions.

Instead of giving a review of what has been written on the most important of the above headings, I have induced several investigators and experts to contribute the results of their work and opinions. If these are fully discussed, and followed up by the results of others who are working in the same line, we shall soon accumulate a large amount of valuable material. That such material already exists there is no reason to doubt.

Several works have employed chemists, and struggled with these problems for years, but each one carefully guards the results; and at the present time many seem disinclined to add theirs to the common store. But, if they could be induced to make their methods and results public, by freely discussing them before this Institute, they would be well repaid by the assistance they would receive through such an interchange. Nobody is as wise as everybody; and nobody loses, as a general rule, by a generous frankness which secures the criticism and help of others. The possible loss through giving to competitors a few shop-secrets is really trivial in comparison.

I am fully convinced that the relations between the chemical constitution and physical character of cast-iron are much closer than is generally admitted to-day. If proof of this is wanted, we have it in the success of those who have given up the old rule-of-thumb methods in running their foundries. There is no doubt that, in a full discussion and investigation of this kind, new data will be brought to light which, in connection with the results of the valuable researches already made, would enable us to make tables, and lay down rules for the founder's guidance, that would be of the greatest value to the makers and users of cast-iron. In other words, from a cupola- or furnace

mixture, of known chemical composition, we could predict with certainty the physical properties of castings of any given size or shape (due allowance being made for change of composition in melting). Or, when certain physical properties were required in a casting of given size and shape, the chemical composition of the different mixtures that would produce these results could at once be given without any of the "cutting and trying" that we now have. This is not too much to expect; and it is not too much to hope that each will do his part in bringing about a result so beneficial to all.

Treatment of Roasted Gold-Ores by Means of Bromine.

BY RICHARD W. LODGE, BOSTON, MASS.

(Florida Meeting, March, 1895.)

Mr. H. R. Batcheller, of the class of 1894, Massachusetts Institute of Technology, while experimenting with chlorine gas on a certain lot of roasted concentrates, met with the following difficulties: 1. A poor extraction of the gold. 2. A very large consumption of chlorine gas. 3. Inability to precipitate all of the gold from the solution containing the $AuCl_3$. 4. The bullion obtained was very base.

These difficulties were the same whether the chlorine was generated from H_2SO_4, MnO_2, and salt, or whether H_2SO_4 and bleaching powder were used. They may be accounted for partly by the presence of some arsenic left in the roasted ore, and partly by the presence of copper in the solution containing the $AuCl_3$.

It was therefore suggested to try the effect of bromine on a similar lot of ore. The use of this element is, of course, nothing new, but in the following experiments it seemed to present many advantages over chlorine.

The material worked upon consisted of some concentrates containing 2.31 ounces of gold per ton, and 34.26 per cent. of arsenic, which would correspond to about 74.4 per cent. of

arsenopyrite. Considerable pyrite and a small amount of galena and chalcopyrite were also present.

The material, when sized and assayed, showed:

	Per cent.	Ozs. gold per ton.
On 24-mesh sieve,	.7	assaying 1.4
" 30 "	1.9	
" 40 "	3.5	" 1.2
" 50 "	6.0	
" 60 "	4.5	" 1.12
" 80 "	11.0	" 1.19
" 100 "	26.0	" 1.4
Through 100 mesh-sieve,	45.0	
Loss,	1.4	
	100.0	

The line of treatment was as follows:

1. Roasting the concentrates in a reverberatory furnace.
2. Submitting the roasted ore to bromination in strong preserve-jars, "lightning brand," with double gaskets, the jars and their contents being revolved during the experiment.
3. Precipitation of the gold by means of H_2S.

Roast I.—Time, five hours.

	Kilos.	Assay. Ozs. gold.
Raw ore,	10	2.31
Roasted ore,	6	3.36
	Per cent.	Per cent.
Loss,	40	12.7

Bromination:

Roasted ore,	500 grammes.
Bromine,	14.5 c.c.
Water,	500 c.c.
Time,	5½ hours.

Assay of tailings from two tests gave 0.30 and 0.32 ounces of gold. Based on the roasted ore, this would be an extraction of 90.7 per cent.

Roast II.—Time, eight hours.

	Kilos.	Assay. Ozs. gold.
Raw ore,	15	2.31
Roasted ore,	8	4.29
	Per cent.	Per cent.
Loss,	46.67	1

The following experiments were made to determine the proper amount of bromine for 500 grammes of ore:

Roasted ore. Grammes.	Bromine. C.C.	Time. Hours.	Water. C.C.	Extraction, based on assay of tailings. Per cent.
500,	3.0	5½	500	90.67
500,	3.0	5½	500	89.27
500,	1.5	5½	500	92.54
500,	1.0	5½	500	81.35
500,	0.5	5½	500	62.23
500,	0.3	5½	500	60.00

The following were made to determine the shortest period of contact of ore and bromine, giving a good extraction:

Roasted ore. Grammes.	Bromine. C.C.	Time. Hours.	Water. C.C.	Extraction, based on tailings. Per cent.
500,	1.5	5½	500	92.54
500,	1.5	4½	500	88.00
500,	1.5	3½	500	86.00
500,	1.5	2	500	81.35
500,	1.5	1	500	72.02

These tests seem to indicate that 1.5 c.c. of bromine, added to 500 grammes of ore in 500 c.c. of water, would effect in five and one-half hours an extraction of over 90 per cent. of the gold in the ore.

To test these conclusions, a third roast was made:

Roast III.—Time, eight hours. Ore cooled in furnace.

	Kilos.	Assay. Ounces.	Arsenic. Per cent.	Sulphur. Per cent.
Raw ore,	70	2.31	34.26	
Roasted ore,	43.7	3.58	0.11	0.34
	Per cent.	Per cent.		
Loss,	37.6	$3\frac{8}{10}$	99.67	

Of this roasted ore 15 kilos were treated with 45 c.c. of bromine in 15 kilos of water for four and one-half hours in a revolving keg. The tailings showed an extraction of 85.5 per cent.

As an excess of bromine was present when the keg was opened, at the end of four and one-half hours, a second experi-

ment was tried with ore, 15 kilos; bromine, 35 c.c.; time, five and one-half hours; water, 15 kilos.

This showed an extraction of 92.18 per cent., based on the assay of the tailings. The actual gold recovered from the solution was only about 80 per cent., which may be accounted for by the presence of considerable copper in the solution.

The expulsion of the bromine from the solution seemed to be best brought about by means of SO_2. Air and steam were both tried, but with poor success. After the passage of SO_2, the solution was quite clear, although some gold would be precipitated on standing.

When the ore was chlorinated, the solution at this point, containing the $AuCl_3$, would be quite turbid, and evidently contained a large amount of base metals as chlorides. These would necessarily interfere with the complete precipitation of the gold, besides making the bullion base. Some base metals, such as copper, were also present in the bromine solution, but apparently not to such an extent; for the solution was clear.

The gold was finally precipitated by means of H_2S.

In the experiments on this particular ore, bromine seemed to have the following advantages over chlorine:

1. It extracted a much higher percentage than chlorine; the results being estimated, not only on the assay of the tailings, but also on the actual gold recovered.

2. It gave solutions much more free from base metals. This would be expected, especially where chlorine is generated by means of H_2SO_4 and bleaching powder, and the acid has a chance to act directly on the ore.

3. Less time is required to extract the gold.

4. The ease in using and comfort in handling is much greater.

As regards the comparative cost, the least amount of bromine which could be used on this ore with a successful extraction, appeared to be 0.3 per cent., or 6 pounds per ton. With bromine at 25 to 40 cents per pound, this would make the cost very high; but the cost of chlorination would certainly be still higher, as it was found necessary to use as high as 10 per cent. of lime and 6 per cent. of H_2SO_4 to obtain even a fair extraction.

The Cyanide Process as Applied to the Concentrates from a Nova Scotia Gold-Ore.

BY RICHARD W. LODGE, BOSTON, MASS.

(Florida Meeting, October, 1895.)

THE following work, performed by Mr. W. A. Tucker, of the class of 1893, in the mining department of the Massachusetts Institute of Technology, seems to me to be worthy of publication. I believe it has always been considered that the presence of arsenic especially interferes with the extraction of gold by the cyanide method. Mr. Tucker's work, although made on a laboratory scale, certainly seems to disprove this view, and to show that even with a very large percentage of arsenic present in the ore, a high extraction may be obtained without an excessive consumption of potassic cyanide.

The ore from which the concentrates were obtained was a gray argillaceous schist and slate, with stringers and veins of quartz running through it. It carried free gold and about 12 per cent. of sulphides. The ore was crushed with stamps; the free gold was collected in the ordinary way on silver-amalgamated copper plates; and the sulphides, which consisted chiefly of arsenopyrite and pyrite, with very small amounts of galena and chalcopyrite, were concentrated and collected by means of a Frue vanner.

A carefully-taken sample gave:

Gold, ...	6.17 ounces per ton.
Arsenic, ...	30.6 per cent.

The latter figure would correspond to about 66.5 per cent. of arsenopyrite in the concentrates.

The work to be done was outlined as follows:

1. Sizing the concentrates;
2. Assaying the different sizings;
3. Treating these different sizings with KCy of different degrees of strength for different periods of time.

Owing partly to the small amount of concentrates found on the 40- and 60-mesh sieves, and partly to the lack of time, the following four series were substituted in place of carrying out No. 3.

Series I.—Treating concentrates with a given amount of a 1 per cent. KCy solution for different periods of time. The solution, instead of being all added at once, was added in three portions.

Series II.—Treating a given amount of concentrates with an equal quantity of a 1 per cent. solution of KCy for different periods of time, the KCy not being renewed as in Series I.

Series III.—The same as Series II., except that the concentrates were revolved with the KCy solution in bottles, and did not simply stand in contact with it, as in the previous series.

Series IV.—Concentrates and solution in motion; strength of KCy solution, time of contact and amount of solution varying.

Sizing and Assaying Concentrates.

A sample of the concentrates sized and assayed resulted as follows:

Sieve-Mesh.		Proportion of Sample. Per cent.	Assay. Ounces per ton.	Gold.	Proportion of Total Gold. Per cent.
Through.	On.				
	40	0.412	25.70	0.000363	1.71
40	60	0.449	27.10	0.000417	1.96
60	80	4.010	10.69	0.001468	6.90
80	....	92.710	6.00	0.019005	89.43
Loss....................................		2.419	....	....	....
Total....................................		100.000	....	0.021253	100.00

The above assays include, of course, the free gold (pellets) which may have been found on the 40-, 60-, and 80-mesh sieves.

Treatment with Cyanide.

Series I.—One A. T., or 29.166 grammes, of concentrates passed through a 30-mesh sieve, and assaying 6.17 ounces per

ton, was treated with 100 c.c. of KCy (1 per cent.) solution. This solution was added at three different times in equal portions; the first portion being drawn off before the second was added, and so on.

The apparatus employed was an inverted glass bottle, with the bottom cut off, and a perforated porcelain plate laid across at the point of contraction to the neck, so as to form (in the inverted position) a false bottom. Below this, the neck was closed with a rubber stopper, through which passed a glass tube, fitting outside to a rubber tube, closed with a pinch-cock.

The result of these tests was as follows:

Time of adding KCy, calculated from first addition.		Time of withdrawing the third addition of KCy.	Assay of tailings in ounces per ton.	Percentage of gold extracted.	Grammes of KCy consumed.	Grammes of KCy used per gramme of gold extracted.
Second.	Third.					
Hours.	Hours.	Hours.				
16¼	20½	23	1.92	68.88	0.112	26.4
16¼	20½	23	2.39	61.26	0.124	32.8
23½	30	51	1.43	76.82	0.168	35.4
22½	29½	51	2.82	54.29	0.146	43.6
23	65	70	1.44	76.66	0.155	32.8
23	65	70	2.78	54.94	0.156	46.0
24	44	94	1.51	75.52	0.169	36.3
24	44	94	2.67	56.72	0.206	58.9
24	44	94	1.51	75.52	0.165	35.4
20½	94½	118	1.62	73.74	0.164	36.0
20½	94½	118	1.56	74.72	0.202	43.8
20½	94½	118	1.23	80.06	0.202	40.9

These results were not at all satisfactory; for they neither indicated that the extraction increased with the time of contact nor did they show in what period of the contact the solution of the gold took place.

Series II.—The apparatus used was the same as in Series I. Quantity of concentrates (through 30-mesh), 25 grammes; assay, 6.17 ounces per ton; quantity of KCy (1 per cent.) solution, 25 c.c. The solution was not changed.

These experiments seem to show that the extraction of gold increases with the time of contact of the KCy. Apparently, the consumption of KCy increases with the time. This large consumption in both Series I. and II. is no doubt due to the free access of air to the apparatus in which the tests were made.

Duration of Treatment.	Assay of Tailings.	Per cent. of Gold extracted.	Grammes of KCy consumed.	Grammes KCy used per gramme of Gold extracted.
Hours.	Ounces per ton.			
16	2.97	51.84	.064	23.4
16	2.75	55.43	.065	22.2
22	2.30	62.72	.058	17.5
22	2.17	64.83	.058	16.9
25¼	2.12	65.64	.065	19.6
71	2.51	59.32	.124	39.5
71	2.24	63.69	.118	35.0
71	2.57	58.35	.127	41.1
118	1.61	73.91	.091	23.3
118	1.40	77.31	.131	32.5
118	1.26	79.58	.134	31.1

While working on these experiments, 25 grammes of concentrates and 25 c.c. of KCy solution were put in a bottle, tightly stoppered, which was caused to revolve. The extraction was such an improvement on all the previous work that all other experiments were conducted in this way.

Series III.—Quantity of concentrates (through 30-mesh), 25 grammes; assay, 6.17 ounces per ton; quantity of KCy (1 per cent.) solution, 25 c.c. Bottles and contents revolved.

Duration of Revo ution.	Assay of Tailings.	Per cent. extracted, calculated from tailings.	Grammes of KCy consumed.	Grammes of KCy used per gramme of Gold extracted.
Hours.	Ounces per ton.			
2	0.39	93.68	0.022	4.44
2	1.11	82.01	0.033	7.60
2	0.82	86.71	0.035	7.63
2	0.82	86.71	0.032	6.97
4	0.66	89.30	0.072	15.25
4	0.62	89.95	0.042	8.82
5¼	0.58	90.60	0.053	11.06
5¼	0.47	92.38	0.063	13.07
23	0.42	93.19	0.031	6.42
23	0.58	90.60	0.043	8.98

These experiments seem to indicate that to revolve the bottles about six hours was sufficient, and that the extra amount of gold extracted would hardly compensate for a longer revolution.

Series IV.—Bottles and contents revolved.

Concentrates Through 30-Mesh; Assay, 6.17 Ounces per Ton.

Duration of Revolution.	Weight of Ore.	Strength of KCy.	Quantity of KCy Solution.	Assay of Tailings.	Per cent. of Gold extracted.	Grammes of KCy consumed.	Grammes of KCy per gramme of Gold extracted.
Hours.	Grammes.	Per cent.	C.c.	Oz. per ton.			
4	50	1.0	25	0.87	85.90	.026	2.84
4	50	1.0	25	0.71	88.49	.094	10.03
4	25	0.5	25	0.47	92.38	.014	2.86
4	25	0.5	25	0.66	89.30	.009	1.91
4	25	0.5	25	0.89	85.57	.014	3.09
16½	25	0.5	25	0.97	84.28	.055	12.33
16½	25	0.5	25	0.57	90.76	.050	10.42
23	25	0.5	25	0.51	91.73	.045	9.28

Concentrates Through 80-Mesh; Assay, 6 Ounces per Ton.

Duration of Revolution.	Weight of Ore.	Strength of KCy.	Quantity of KCy Solution.	Assay of Tailings.	Per cent. of Gold extracted.	Grammes of KCy consumed.	Grammes of KCy per gramme of Gold extracted.
Hours.	Grammes.	Per cent.	C.c.	Oz. per ton.			
6¾	1000	1.0	1000	0.70	88.30	7.033	77.06
6½	1000	0.5	1000	1.00	83.33	3.617	40.22

The large consumption of KCy in these last two tests was due to insufficient washing.

In none of the tests were the concentrates washed with water previous to their treatment with cyanide. Owing to lack of time, Mr. Tucker was unable to test the solutions for arsenic, or to see whether all the gold could be recovered from them; so we are unable to give any data on these points. While we realize that these are simply laboratory experiments, that the tailings are in all cases too rich to be thrown away, still we consider the extraction remarkably high on material carrying the percentage of arsenic that this does. Making the tests in a closed vessel lessens the consumption of KCy, as one would expect. As the extraction also increases, this would seem to be contrary to Elsner's equation, to which oxygen is necessary; and certainly there could hardly be enough in a small bottle to influence the extraction. Keeping the ore and solution in agitation certainly seems to have helped the extraction, although this method of working has met with very little success in actual practice.

A New Slag-Car for Lead and Copper Blast-Furnaces.

BY CARL HENRICH, DUCKTOWN, TENN.

(Florida Meeting, March, 1895.)

WHILE the size of the blast-furnaces used for smelting lead-nd copper-ores has constantly increased, during late years, the nanner of removing the slag from the furnace to the slag-dump ias (until quite recently, at least) practically remained the same. The only appliance in practical use has been the two-wheeled lag-pot, holding about 300 to 400 pounds of slag, or as much s one man could manage to pull or push on a smooth, hard urface. When these slag-pots on wheels were first introduced, he daily capacity of a lead or copper blast-furnace hardly eached 30 tons of smelting-charge. With furnaces of this ize, and even when the capacity had been doubled, this kind f slag-pot answered well enough. All the improvements made vere in the *construction* of these pots and not in an increase of heir size, which was limited by the weight which one man ould handle.

But the size of the furnace went on increasing. The daily onsumption of one furnace rose gradually to 75 tons, 100 ons, and even more. The writer had occasion to construct ome of these furnaces, which, in their day, were among the argest, if not the largest, of the water-jacketed kind then in xistence. A water-jacketed furnace will always smelt more n the same time than a furnace with brick walls.

Already, in those days, the slag-removal from these enlarged urnaces received due attention by the writer. The removal of 60 tons of slag in 24 hours, in pots holding only 300 pounds ach, will require 400 pots to be filled in 24 hours; and when he amount of slag increases to 90 tons, 600 pots will be re-quired. That is to say, in the last case, 25 pots in each hour, r 1 pot each $2\frac{1}{2}$ minutes, will have to be put under the slag-pout, filled, pulled away, and replaced with another. This will nvolve too much hurry for any degree of comfort, convenience

or cleanliness, and will surely not be conducive to good work on the part of the men engaged in the hot, disagreeable, and even dangerous business.

Providing two slag-taps, one at each end of the long rectangular furnace, will remedy, to some extent, this congested state of affairs; but it will not diminish the excessive amount of human labor, and the large number of pots needed for the removal of the slag, especially when the distance from the furnace to the edge of the slag-dump increases as it does, frequently more rapidly than is expected or desired.

But furnaces capable of smelting 200 tons daily of roasted pyritic ores are now constructed, and can be run as successfully as smaller ones, and at less cost per ton of ore smelted. To remove the slag from a furnace of this capacity by the use of the old-fashioned two-wheeled slag-pot would result in an intolerably congested state in front, or even at both ends, of such a furnace.

When the writer, therefore, constructed a furnace of this capacity (or of larger capacity, when driven) for the smelting of the roasted pyrrhotite-ores of Ducktown, he realized that, with this increase in the size of the furnace, a new way of removing the slag would have to go hand in hand. We might as well attempt to remove the cinder of an iron blast-furnace in two-wheeled slag-pots by hand, as to do this for a lead- or copper-furnace of 200 tons' daily capacity.

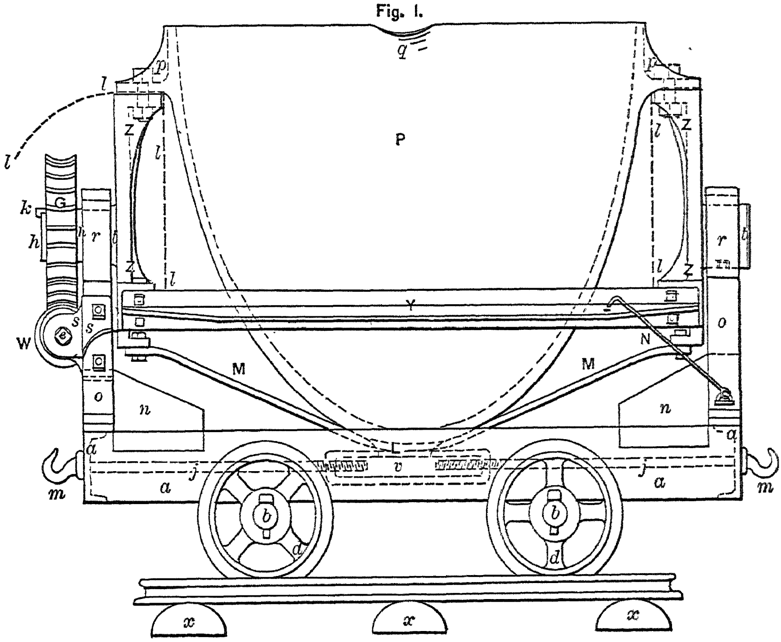
The cinder of an iron-furnace may be tapped into slag-cars standing on a tramway below the cinder-notch. Why not do the same at a copper-furnace? But the nature of the slag is different, and the shape of the slag-car and the manner of emptying it had to be modified. The outcome of much study of the subject was the slag-car shown in Figs. 1 and 2, which was designed by the writer, and provided with the worm-gear turning-device (in place of a single lever) by Mr. James Scott, superintendent of the Lucy Furnaces, in Pittsburgh, to whom the original plan was submitted.

Fig. 1 is a side-elevation, and Fig. 2 an end-elevation of this car, as constructed for the copper-furnace of the Polk County Smelting Works, at Ducktown, Tennessee.

This furnace has been built for smelting the roasted pyrrhotite-ore of the Polk County mine, together with the slaty

and quartzy ore from the same mine (also roasted), in such proportions as to make an easy-melting mixture. The capacity of the furnace, with a blast-pressure of 6 to 7 ounces per square inch (the blast supplied through 16 four-inch tuyeres), was estimated by the writer at about 160 to 180 tons in 24 hours. A trial-run of 1000 tons of roasted ore showed, however, that the capacity of the furnace was larger, and, with a perfectly clean furnace and a normal running condition, would easily reach

Fig. 1.



SLAG-CAR. SIDE ELEVATION.
Scale 1½″=1′=⅛ Nat. Size.

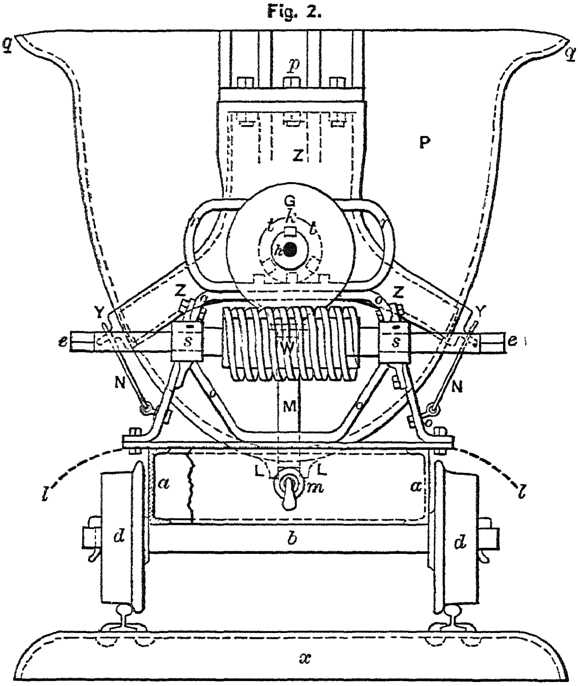
180 to 200 tons, and even more, with the blast-pressure mentioned and a full supply of blast.

With such a capacity of the furnace, 180 tons of slag in 24 hours, or 15,000 pounds of slag in an hour, will, therefore, be only a normal performance. But, as these pots or cars hold 5000 to 5500 pounds of slag, it will be necessary only to change these slag-pots every 20 minutes. This allows the stopping of the slag-hole and the tapping of the matte, in the intervals between two consecutive slag-pots. Instead of the rush and annoyance of changing 40 to 50 small slag-pots every hour, with the constant necessity of cleaning up the spilled slag from the

floor, the work becomes convenient, easy, and comfortable. Other advantages, inherent in the large capacity of the slag-car, will be mentioned later on.

Figs. 1 and 2 need little explanation. The pot, P, is made of cast-iron, and is here shown as elliptical on top (42 by 48 inches) with a lip, *q*, protruding an additional 6 inches on either side. It will probably be better to make the horizontal sections of the pot true circles, say 48 inches in diameter in this case on

Fig. 2.



SLAG-CAR. END ELEVATION.
AT END WHERE WORM WHEEL GEAR IS PLACED.
Scale 1¼″ = 1′ = $\frac{1}{16}$ Nat. Size.

the top, not taking the lips into consideration. It will also be probably better to make the vertical section of the pot more in the shape of a cone, instead of egg-shaped, or spheroidal, as in the figures. The conical shape, or any shape approaching the cone, will facilitate the sliding-out of the skull, after the pot has been emptied. The depth of the pot is a little over 3 feet. This gives *a working-load* of melted slag of about 25 cubic feet.

At the outside of the pot, just below the upper rim and half-way between the lips, two stout brackets, *p*, are cast to the pot.

Through these brackets, which rest on the car frame, the pot is bolted to the trunnion-castings, Z, one at each end of the slag-car.

These trunnion-castings are of the peculiar shape shown in the sketches. They carry the trunnions, *t*, on which the pot turns vertically. And they have four arms. One of these, the largest and stoutest one, supports the pot P, being bolted to the brackets, *p*. Two of the arms of each trunnion-casting, Z, are joined by T-irons, Y, to the corresponding arms of the opposite casting. The fourth and lower arms are joined by a heavy piece of flat iron, M, which runs underneath the pot, P, between lugs, L, cast on the outside of the bottom of the pot. This flat iron serves to support and steady the pot, and also serves as a tension-rod, when the additional weight of the liquid slag makes such a tension-rod in the frame-construction desirable.

On a narrower prolongation, *h*, of one of the trunnions, *t*, the worm-gear wheel, G, is fastened by the key, *k*. The trunnions, *t*, *roll* in the frames, *r*, made of heavy flat iron. These frames, *r*, are supported by other upright frame-supports, *o*, likewise of heavy flat iron, all riveted and bolted together, and bolted to the car-body, or frame, *a*, which is made of 4 pieces of channel-iron, fastened together at the corners by angle-iron. To this rigid car-body the vertical frames, *o*, are bolted, and held in vertical position by the steel plates, *n*, put at the four corners.

To the frame, *o*, at one end of the car, are also bolted, at the proper height, the box-castings, *s*, in which the axle of the worm, W, turns. This axle, *e*, *e*, has squared ends, to which a properly crooked crank fits, by which one man is enabled to tip the empty, or the full pot, and pour the contents, quickly or slowly, as may be desired. In fact, if the trunnions are put at the proper height, a small child would have sufficient strength for the work, many times over.

Sheet-steel aprons, *l*, protect the wheels, *d*, and the bearings of the axles, *b*, from the splashing of the molten slag. Sheets of steel, *l*, of sufficient thickness to be stiff and not easily bent out of shape, are put vertically between the slag-pot, P, and the trunnion-castings, Z, to protect the trunnions and the gear, as much as possible, from the heat radiating from the pot. Another apron, *l*, extends over the gear-wheel, to prevent any splash-

ing of molten slag into it. These latter sheets are fastened between the brackets, *p*, and the top of the trunnion-castings, Z.

Hooks, *m*, are fastened to the ends of the car-body, and are connected with each other by the stout rod, *j*, and the swivel-joint, *r*, with right- and left-hand threads, to tighten the rod. These hooks serve, of course, as attachments of the motive power, mule or locomotive, and for coupling several cars together, as would be desirable, if the slag had to be hauled to any considerable distance by locomotive.

These cars run on light T-rails. At the Polk County works these rails were fastened to common oak ties, for the sake of saving first cost of plant. It will be advisable, however, to use the hollow steel ties, *x*, made of heavy pressed steel plate, which are shown in Figs. 1 and 2, and to which the steel rails are fastened by riveting. Wooden ties burn out too quickly to be really economical.

The thickness of the cast-iron of the pot, P, is $\frac{5}{8}$ to $\frac{3}{4}$ inch at the upper rim, and $1\frac{1}{4}$ inches at the bottom.

The writer is satisfied that such large cast-iron pots will stand wear and tear at a furnace much longer than the small pots now in use. It takes a little over one minute to fill a small pot, and the temperature of the pot-casting is thereby subjected to much more sudden changes, than with these large pots, which it takes 20 minutes to fill with liquid slag. Six of these large pots, holding 5000 pounds slags each, to one furnace making 15,000 pounds of slag per hour, will allow 20 to 30 minutes for heating the pot, and then about $1\frac{1}{2}$ hours for cooling, after emptying it, before it will be filled and heated again.

With the old slag-pots, holding, say 375 pounds slag each (which is too large an estimate for working-weight), it would take from 66 to 75 slag-pots in constant regular rotation to allow them to cool $1\frac{1}{2}$ hours before using them again. With less than 36 to 40 constantly in use and in rotation, the slag-pots would gradually get too hot for use, as there would not be sufficient time to cool them, even if the liquid contents should be emptied as soon as practically possible.

Owing to the large size of this slag-pot, it acts as an additional settler. If any "prills" of matte escape from the sump or fore-hearth with the slag, they will have time, during

the 20 minutes, which it takes (even with rapid running of the largest furnace) to fill one of these pots, to settle to the bottom of the pot and there become fastened in the skull, which forms all over the perfectly cooled slag-pot. The investigation of the bottoms of these skulls is a very good indicator of "prills" of matte in the slag.

Owing to the large volume of slag contained in one of these pots,* the slag remains much longer liquid than in a small pot; and the volume of the skulls is also much smaller for the same amount of slag. This last is a decided advantage, when it is desired to remelt skulls on account of valuable matte contained therein.

That the labor involved in removing, at the most, 3 of these pots per hour to the slag-dump, is much smaller, or much less expensive, than the labor of removing in the same time 80 small pots, is self-evident. One mule and driver are amply able to take care of that quantity of slag in these large cars. If more than one furnace is to be served, the expense will be still further reduced by employing a light locomotive.

It appeared, at first thought, to the writer, that the moving of the railroad-track, as the edge of the slag-dump is extended, might be a serious item of expense by this method. But the slag in these pots stays perfectly liquid so long, that it spreads out on a very gentle slope when dumped; and the movement of the track farther out over the edge of the slag-dump can be done for considerable distances at one time, and need be done, therefore, only at long intervals. It will be found in practice that the expense of moving the slag-car tracks will be so little, that it need not be considered at all, at least not with such slags as are usually produced at copper- and lead-smelting works.

While these cars have not yet had any long practical trial, their behavior during the above-mentioned trial-run of 1000 tons of ore (the only smelting done so far at these works) has been such as to warrant pronouncing them a complete success.

There is no patent, either on the design or on any feature of these cars; and every one is welcome to use them in their present shape or modified to suit other conditions.

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* About 16 times as much slag, and only about 6 or 7 times as much cooling surface as in a small slag-pot.

The floor of the furnace-room in front of the slag-tap, or at least the grade of the slag-track, must be made considerably lower, than is now the custom. But this can be easily arranged.

If the steadying-hooks, N, shown in the sketches, or any other appliance, be used, to prevent undesirable tipping of the pot, after the worm-gear gets "limbered up" through wear, these cars might be taken by locomotive- or mule-power to any desired point at a distance, where the slag could be utilized as a road- or foundation-material, and there dumped without rehandling. Other conveniences may be found in the wake of the use of this car; but its chief merit will be the relief it will bring to the congested condition now prevailing around the slag-tap of every modern large copper or lead blast-furnace, and the saving in the expense of removing the slag.

It is the writer's belief, that furnaces would have been made much larger in lead- and copper-smelting long ago, if the difficulty of handling the slag had not stood in the way.

The Present Limitations of the Cyanide Process.

BY C. W. MERRILL, B.S., SAN FRANCISCO, CAL.

(Florida Meeting, March, 1895.)

THE cyanide process in the United States, notwithstanding numerous failures made under the direction of the owners of the patent-rights, and others, has now passed its experimental stage, and can, undoubtedly, be made successful when intelligently applied to suitable material.

Much, however, remains to be known as to the exact conditions under which this process is applicable to the treatment of gold-bearing ores. If the owners of the patent-rights in this country have ever scientifically investigated the matter, they have never published any valuable information on the subject. Other parties who are now using the process (often independently of the patentees), have nothing to gain by publishing their experience; and, moreover, many of the chemical reactions described in the text-books refer to the use of solutions of potassium cyanide many times stronger than are used in the metallurgical process.

I have, therefore, ventured to outline in the following paper some conclusions from laboratory- and field-investigations, hoping thereby to induce others to communicate their knowledge, and thus to give new light upon certain points which are as yet unsettled; also, to call attention to matters which should be kept in view in considering the treatment of ores by this method; and, finally, to add what little I can to the knowledge now extant upon the subject.

Unless otherwise specified, the term ore, as used in this paper, will signify either ore or tailings.

The consideration of ores of which the silver forms an important factor of their value, will be omitted, although there are good reasons for believing that, with some modifications, certain silver and silver-gold ores can be successfully treated.

Ores containing coarse gold, which is not easily soluble in potassium cyanide, should always have that portion of the gold extracted by concentration or amalgamation before leaching. This precaution being observed, out of some fifteen gold-ores on which I have experimented, not one has been found in which more than 25 per cent. of the original value was left in the tailings after leaching with a solution of potassium cyanide. In some cases less than 5 per cent. so remained; the average being about 15 per cent. But often the decomposition of cyanide of potassium was so great as to prohibit the application of the process. The assertion of the patentees, which forms the basis of their original United States patent, to the effect that solutions containing cyanogen in a proportion not exceeding 8 parts of cyanogen to 1000 parts of water (which is equivalent to a 2 per cent. solution of potassium cyanide), will dissolve gold and silver in the ore, while leaving the base metals practically intact, is contrary to the fact, as a solution containing as little as one-half of 1 per cent. or less of potassium cyanide, will dissolve compounds of some of the base metals, and especially of copper, zinc, and manganese, with the greatest facility.

Of the ores unsuited to this process, some are so for chemical reasons. Of these, cupriferous ores are specially to be mentioned. The great affinity of all copper minerals (save possibly the silicate, the phosphate, and native copper) for cyanogen, is the cause of a large consumption of cyanide of potassium, which is an expensive reagent. A means of recovering the

cyanogen thus rendered useless has not as yet been devised. A few figures will illustrate this large consumption. One pound of copper will combine with a little more than two pounds of potassium cyanide to form the double cyanide of copper and potassium; therefore, if an ore contains as little as one-half of 1 per cent. or ten pounds per ton of copper, in a mineral form which is soluble in the solution used, the amount of cyanide of potassium rendered useless will be more than twenty pounds, costing, at fifty cents a pound, more than ten dollars.

The same line of reasoning applies to oxidized zinc-bearing ores, but the affinity for cyanogen does not seem to be as great in zinc-minerals as in those of copper.

The consumption of potassium cyanide occasioned by the presence of certain compounds of iron and manganese can sometimes be obviated, to a great extent, by mixing lime with the ore; but at other times there is, in leaching partially oxidized pyritic ore, a consumption of potassium cyanide which the use of lime will not overcome. There is no precise information available as to the mineralogical or chemical conditions under which these differences occur.

Other ores offer difficulties because of their physical nature. This is the case with ores naturally soft or slimy, and yet not porous enough to give a good extraction when crushed coarse. By fine crushing these are rendered so impermeable to the solution that successful leaching is impossible.

Similarly, the treatment of slimes which have become segregated from the sandy constituents of the ore has not, as yet, been solved in a practical manner.*

Furthermore, there is a class of roasted material, in the roasting of which the temperature was so high as to fuse the minerals in which the gold occurs, thus so imprisoning the gold as to render it incapable of being dissolved by the cyanide solution. Or, possibly, the physical condition of the gold itself has been so altered as to render it insoluble in the solution.

Coming now to the, as yet, undetermined class of ores, trust-

* Since writing the above, I have made experiments, adding 10 per cent. of slimes (which had been disintegrated to pass a 4-mesh screen) to ordinary tailings. The results were very successful; and a 150-ton plant is under construction, to work on that basis. Furthermore, Mr. Butters has apparently solved the problem of treating pure slimes; but details of his method are not yet available.

worthy information from continuous working of large quantities is still lacking in regard to the behavior toward cyanide solutions of ores containing galena, zinc-blende, or oxidized lead compounds, also of telluride ores and unoxidized pyritic ores. Their adaptability to the treatment will depend, in the first place, upon the condition in which the gold is present (whether distinct from the base metals or chemically or intimately combined with the latter), and, in the second place, upon the solubility of the base metals in a solution of potassium cyanide of a strength sufficient to extract the gold. Where this solubility exists to any great extent the consumption of potassium cyanide will render the process unavailable. Moreover, the solubility of the base metals entails other evil effects besides the chemical loss of potassium cyanide. As the solution becomes fouled by their presence, its extractive power becomes weakened, even though the normal percentage of active cyanogen be maintained by the addition of fresh quantities of potassium cyanide.* Again, the gold and silver contained in solutions of this kind are imperfectly precipitated in contact with metallic zinc. The zinc filaments used in practice for precipitating the precious metals become incrusted with a deposit which prevents their further action to such an extent that, as has happened in a large plant erected for the cyanide process, the solution, after it has been used on the ore a few times, will flow out of the zinc boxes almost as rich as it went in, causing, with other difficulties in the treatment, the failure of the process at this place.

The class of ores suitable for this process comprises all oxidized ores which are not included in the above classes; also, roasted tailings from which the coarse gold has been removed, and which do not contain such compounds of the base metals as will consume too large a quantity of potassium cyanide; and, finally, strictly quartzose or siliceous ores in which a considerable quantity of fine gold remains after amalgamation. A small percentage of pyrites in such cases will not prove an insurmountable objection.

* As a possible explanation of this fact, I take pleasure in calling attention to Mr. Maclaurin's paper in the *Journal of the Chemical Society* for February, 1895, p. 212, in which he clearly and convincingly shows that the rate of dissolution of gold in cyanide solutions decreases as the viscosity of such solutions increases, and as the absorption-coefficient of oxygen decreases.

In conclusion, it may be said that, as far as known at present, only those ores can be economically treated which will readily yield a fair percentage of their gold-contents to the action of weak solutions of potassium cyanide, and which contain no appreciable quantity of base-metal compounds soluble in such weak solutions. While strong solutions will extract a high percentage of the value from a much larger class of ores, no method has as yet been made known for overcoming the excessive chemical consumption and the consequent fouling of the solution accompanying their use.

In deciding upon the application of the cyanide process to the treatment of a new ore, it is, therefore, not sufficient to know, in a general way, that a high percentage of the gold is soluble in potassium cyanide. The minimum strength of solution which will be required to obtain satisfactory results, and the unavoidable loss of potassium cyanide by chemical decomposition, should be carefully ascertained by laboratory-tests, as well as the permeability of the crushed ore to the leaching-solution. An investigation should also be made to determine the ability of zinc to precipitate the precious metals after continued use of both the solution and the zinc.

Mining Leases.

BY FRANCIS T. FREELAND, ASPEN, COLO.

(Florida Meeting, March, 1895.)

In the West many precious-metal mines are worked in patches by lessees, under conditions which closely resemble those of what is called "tribute-work" in Cornwall.* The company has its own foreman or inspector, and usually employs the engineman, blacksmith, pumpman, and perhaps other general workmen.

As an inducement to work on a new piece of ground, where no ore is in sight, the company may advance a "footage"

* I believe the only reference to this subject in the *Transactions*, is the paper of Mr. B. B. Lawrence, *Trans.*, xxi., 911, on "The Lease or Tribute-System of Mining as Practiced in Colorado."

amounting to one-third or one-quarter of a fair contract-price. This enables a miner without capital to pay supply- and board-bills, while the ground is being opened up by an extension of the main level or other working on the vein. If the block becomes productive, the loan will be repaid, otherwise it is commonly lost.

For the same purpose, the company may, in some cases, take an interest in its own lease. For instance, where the company takes a quarter with three working miners, it will advance the wages of one man in cash. Then, when ore is shipped, after taking out its royalties, it will take in addition one-quarter of the net returns to the lessees.

Frequently it will be found advantageous to furnish the lessees with rails and timbers for the main levels, and to sink the main-shaft on company-account, in order to secure adequate workmanship. A certain amount of development-work is required of the lessees, according to the size and character of the block of ground. Such leases are commonly given for six months, and on a strip of the vein 150 feet in length and 75 feet high, or from level to level.

The company sells the ore. For convenience, the royalty is calculated from the returns as given in the settlement-sheet of the ore-buyer. From the share due the lessees is deducted the haulage, tramming, hoisting, sharpening, timber, supplies, and other charges, if any. The company often advances the supplies at cost, and does the hoisting and pumping. If the operation turns out unprofitable, the company loses the charges.

Leases can be granted by selection or competition. Bids may be asked on the royalty; or the royalty may be fixed and a bonus asked, part cash in advance and part to be taken out of the returns in addition to the royalty. I have known a choice of ground to be given in return for the sinking of a shaft. The management will find it better, on the whole, to arrange the rate so that some of the lessees will make more than wages; for there will then be little difficulty in finding takers for other blocks of ground.

This method of working is especially suited to narrow veins of good grade, where careful placing of the shots is required to keep the ore clean and to save it all. It would also have advantages in a district controlled by a strong labor-organization.

As the workmen are interested in the product, they require less watching; no unnecessary dead work is done and the minimum amount of supplies is consumed.

The lessees are made responsible for accidents to their employees, and cannot claim damages against the lessor for personal injuries.*

With careful inspection, many mines can be systematically opened in this way, with profit to their owners, or small danger of heavy losses, and yet with satisfaction to the miners. If the leases are set with judgment and fairness, a reliable set of good miners may be kept about the mine.

A lease of this character must be rigorous in its terms in order to permit effective control, and it will require a special form. I have drawn the blank below for the Pontiac, Champion-Empire, St. Joe and Mineral Farm consolidated companies, operating in Aspen through the Cowenhoven tunnel. It can be readily adapted to other conditions.

Form of Lease.

[After the usual preliminary description of the parties, and the clauses granting and defining the ground leased, "for and in consideration of the royalties, covenants and agreements hereinafter reserved and by the said lessees to be paid, kept and performed," and "for the purpose of mining and for no other purpose," and fixing the term, "unless the same shall be sooner terminated by the violation of any covenant," etc., the agreement continues:]

And, in consideration of such demise, the said lessees do covenant and agree with said Company as follows, to-wit:

1. To enter upon said demised premises and work the same in good and miner-like fashion and in manner necessary to good and economical mining, so as to

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* In Colorado, a verdict for damages in case of death is limited by statute to $5000; but there is no limitation with regard to injury or mutilation not fatal. In most cases, employees would probably find it very difficult to collect from lessees judgments for heavy damages. It is well understood that lessees take greater risks than a company would dare to assume. Much of the economy of lease-work, apart from the scanty amount of development or other dead-work which it includes, lies in the employment of light timbering, insufficient for permanent workings, and often enhancing the immediate risk of accident.

At one time, a Colorado statute required the posting of a notice at the mine, in order to relieve the lessor from liability for wages due and bills contracted by the lessees on account of the lease; but this statute has been repealed. I know of no successful attempts in Colorado to collect such debts from the lessor. Those who furnish supplies to lessees usually require a guaranty from a responsible party. The reservation by the lessor of the right of property in the ore prevents its attachment by creditors of the lessees

take out the greatest amount of ore possible, with due regard to the safety, development and preservation of said premises.

2. To work said premises in the manner aforesaid steadily and continuously from the date of this lease with at least two (2) men or fifty (50) shifts each month, and any failure so to do for the period of three (3) days in any one calendar month, shall work a forfeiture of this agreement without notice.

3. To do no underhand stoping below the bottom of any main working level or winze, and to well and sufficiently timber the workings on said premises at all proper points, and to repair all old timbering whenever it may be necessary, so as to ensure the permanency of the said workings, as may be directed by the agents of said Company.

4. To make all raises and winzes at least three and one-half (3½) by seven (7) feet, and all drifts at least three and one-half (3½) by six (6) feet in the clear. To make main levels for the passage of tunnel cars four (4) by six and one-fourth (6¼) feet in the clear; said Company reserving the right to furnish timbers and rails for the same, said lessees to frame timbers and lay track.

5. To sink, drive or raise certain workings a distance of fifteen (15) feet per month, and to substantially timber the same where necessary, at their own expense and at the designation and under the direction of the agents of said Company, to-wit: . . . .

6. Said Company agrees to loan said lessees the sum of ——— dollars per foot of distance said working may be extended in each calendar month and approved by the agents of said Company, the said sum to be withheld from the portion of the proceeds of any ores mined from said premises due said lessees. Said Company reserves the right to refuse further loans at any time, without notice.

7. All track and loading chutes constructed by said lessees shall be allowed to remain at the termination of this lease as the property of said Company.

8. Said Company reserves a right of way for all purposes jointly or in common with said lessees through all workings made or to be made within said premises.

9. To keep at all times all shafts, drifts, tunnels, raises, winzes, passages and other workings thoroughly drained and clear of loose rock and rubbish, unless prevented by extraordinary mining casualty, and not to obstruct the main openings in any manner whatever. To stow no waste underground, except with the consent and under the direction of the agents of said Company.

10. To preserve a due regard for the rights and convenience of other workmen and lessees using jointly or in common the same main openings, track, cars, hoisting and ventilating machinery while tramming, shooting and working.

11. In any case where the removal of timbering or breaking of ground would injure the permanency of the workings, the said Company, by its agents, shall have the right to prohibit such operations.

12. Not to mine outside the boundaries of said demised premises and within the property of said Company or within territory and lodes not leased to them, and said lessees hereby waive any claim to or interest in the proceeds of ore taken in such manner or for compensation for mining the same.

13. To allow the agents or attorneys of said Company to have at all times access to all parts of said premises for the purpose of inspecting, surveying or sampling the same. To allow the lessees of the adjoining blocks of ground, upon a written order from the agents of said Company, access to the said premises for the purpose of inspecting the same; and said lessees shall have the privilege, upon a written order from the agents of said Company, of examining the blocks of ground adjoining the said demised premises, in consideration there-

for waiving all claim for damages against the lessees of said adjoining blocks for trespass.

14. Not to mix or adulterate any ores broken or mined, without consent of the agents of said Company. All ores mined which are too low grade for present shipment shall remain the property of and subject to the control and disposition of said Company. Said lessees shall be chargeable with, and pay, any loss or expense resulting from a shipment of ore which may not be of a paying or salable grade.

15. If said premises are worked from the main levels and inclines of said mine, and said Company hoists the ore and waste therefrom and disposes of the same at the surface in bins or on the dumps at its expense, then said lessees shall be chargeable with and pay therefor, —— cents per ton of ore and —— cents per mine car of waste; and if said Company hoists or lowers ore and waste and delivers the same to the Cowenhoven tunnel, then said lessees shall be charged and pay —— cents per ton of ore and —— cents per mine car of waste; and said lessees further agree to pay for tunnel-haulage —— cents per ton of ore and —— cents per mine car of waste; said hoisting, lowering and tunnel-hauling charges to be paid on or before the fifth (5) day of each month for the work done during the preceding calendar month; *provided, however*, that said lessees shall do the underground tramming and waive all claim to or interest in the waste or its final disposition; but it is expressly understood that in case of accident to the machinery or workings, or obstruction to their use by fire, riot or legal process, said Company shall not be compelled to remove such material nor be liable for damages for such failure.

16. It is expressly understood and agreed, that said Company reserves the property and right of property in and to all ores extracted from said premises during the period of this lease.

17. It is further agreed that said Company shall retain as royalty the following percentages of the net sampler- or smelter-returns, reasonable sampling, freight and treatment charges only being first deducted, according to the assay-value thereof in ounces of silver per ton, to-wit:

Royalties.

On net returns.	Silver per ton.
Fifteen (15) per cent.,	Under 25 ounces.
Twenty (20) per cent.,	From 25 to 30 ounces.
Twenty-five (25) per cent.,	From 30 to 40 ounces.
Thirty (30) per cent.,	From 40 to 50 ounces.
Thirty-five (35) per cent.,	From 50 to 75 ounces.
Forty (40) per cent.,	From 75 to 100 ounces.
Forty-five (45) per cent.,	From 100 to 200 ounces.
Fifty (50) per cent.,	200 ounces and over.

18. Said Company shall ship said ore to the Taylor & Brunton Sampling Works, Aspen, Colo., there to be sampled, and sell the same to the highest and best bidder for cash. Said lessees shall have the right to be present or represented at the sampling of said ore, should they so desire. Said lessees shall pay the costs of assaying and of the inspection of the sampling of lots of ore sold; and also the costs of any surveys desired by them.

19. Said Company, on receipt of said net returns, shall deduct therefrom its royalties, calculated by the schedule hereinabove set forth, together with said

hoisting, lowering and tunnel charges due, and other charges and loans, if any there be, and pay the remainder thereof to said lessees on or before the tenth (10) day of the month succeeding the calendar month in which the said ore shall have been sold. Said Company shall not be liable for the proceeds of ore lost by theft, accident or failure of the ore-buyer, not arising from its negligence; nor shall it collect any royalties from ore so lost.

20. And it is hereby agreed by the parties hereto that ——— shall be the agent of said lessees to make settlements, receive moneys from all ores shipped, and to divide the same among his co-partners.

21. Said lessees hereby assume all responsibility, in case of accidents to themselves or any of their employees in or upon the property of said Company.

22. Said lessees or either of them, shall not take to said demised premises or elsewhere for use thereupon, any tools, supplies or other property of said Company, or of other lessees of said Company, without express permission.

23. Not to assign or sub-let this lease or any interest therein, or the premises affected thereby, or any portion thereof, without the written consent of the Company; nor to allow any person not in privity with the parties hereto to take or hold said premises, or any part thereof, under any pretence whatever.

24. To occupy and hold all cross or parallel lodes, spurs or mineral deposits of any kind which may be discovered by the said lessees, or any other person under them, in any manner within the said premises as the property of said Company, with privilege to said lessees of working the same as part of the said demised premises.

25. To deliver to said Company the said premises, with the appurtenances and improvements thereon, in good order and condition, and all shafts, drifts, tunnels, raises, winzes, passages and other workings thoroughly clear of all loose rock and rubbish and drained and ready for immediate and continuous working, accidents not arising from negligence alone excepted, without demand or further notice, on the last day of the term hereof, or at any time previous, upon demand for forfeiture.

26. And finally, that upon the violation of any covenant or covenants hereinbefore reserved, the term of this lease shall, at the option of said Company, expire and the same and the said premises become forfeited without notice to said Company, its successors and assigns, and said Company, its agents or attorneys, may thereupon, after verbal or written demand for possession given to said lessees or any one of them, or mailed to them or any of them, at Aspen, Colorado, or given to any person working under said lessees, enter upon said premises and dispossess all persons occupying the same, with or without force and with or without process of law; or, at the option of said Company said lessees and all persons found in occupation may be proceeded against as guilty of unlawful detainer; and failure by said Company, through its agents, to exercise for any length of time any right of forfeiture for any cause shall in no event operate as a waiver of such right of forfeiture.

I am using a similar form of patch lease on the Durant, Compromise, and Late Acquisition mining companies at Aspen, operating through the Durant tunnel, on the following scale of royalties, and these leases brought bonuses of from $100.00 to $500.00 for six months time in addition to the royalties, one-third cash in advance:

Royalties.

On net returns.	Silver, ounces per ton.
Ten (10) per cent.,	under 20
Fifteen (15) per cent.,	from 20 to 25
Twenty (20) per cent.,	from 25 to 30
Twenty-five (25) per cent.,	from 30 to 40
Thirty (30) per cent.,	from 40 to 50
Thirty-five (35) per cent.,	from 50 to 60
Forty (40) per cent.,	from 60 to 80
Fifty (50) per cent.,	from 80 to 100
Sixty (60) per cent.,	from 100 to 250
Seventy (70) per cent.,	Over 250

The form shown is in marked contrast to the long-time, low-royalty, large-territory leases given to responsible, experienced mine-operators by owners who may be non-resident, or who do not wish to undertake the risks of mining and the cares of management. For this purpose the short form generally on sale by law-stationers will be used, or, in important cases, a special agreement will be drawn. The notable difference in the royalties is due to the large amount of dead work and surface-improvements required in new and extensive enterprises. Here the reputation for honesty, skill, proposed rate of working, financial resources, or backing of the intending operator usually influences the selection. To avoid the cost of employing a local representative, the owner often appoints the ore-buyer as his agent. The buyer will be furnished with the scale, and will send the royalty direct to the owner, with a duplicate settlement-sheet.

Such an arrangement is often better than a purchase, in many ways and for both parties. Some very successful mining companies have been formed upon such leases.

I know of cases where a disinterested party was able to lease ground in litigation from the several claimants, dividing the royalties paid in an agreed proportion between the claimants, and thus terminating the litigation. A further element of success is, that this plan will enable a large territory to be controlled without paying out great sums for purchase, which might cripple the enterprise. The economies resulting from a single management, deep shafts provided with adequate hoisting and pumping machinery, and a systematic planning of the prospecting, development and stoping work, and draining system, are obvious.

Notes on a Southern Coal-Washing Plant.

BY J. J. ORMSBEE, TRACY CITY, TENN.

(Florida Meeting, March, 1895).

ATTEMPTS at coal-washing have been made in the southern states during the last twenty years; but it is only within the last four or five years that the practice has become at all general. It might perhaps be claimed as one of the blessings derived from our departed "booms;" for, during their sway, the supply of coal of all qualities, good and bad, could not equal the demand; but, with the subsidence of the inflated demand, came imperative calls for fuels of better quality; and washers, previously regarded as luxuries, became necessities.

Among those now in use in this section are representatives of the following types or classes: the trough washer; the jig washer; the percussive table; and those washers in which a constant upward current of water effects the separation. Without having full statistics, it is safe to say that there are in successful operation in the South more washers of the last class than of any of the others. The purpose of these notes is to present data with regard to the construction, operation, and results of one of these current-washers, based mainly on the plant at No. 2 Slope, Pratt Mines, Alabama.

The coal is mined from the well-known Pratt seam, having here an average thickness of 3 feet 6 inches. It has distinct cleavage-planes, and breaks in cuboidal lumps; is bright black in color, firm in structure, and air-slacks only after considerable exposure. It burns freely, leaving a gray or buff-colored ash. The lump- and nut-coals are used for domestic and steam-purposes (chiefly, however, for locomotive firing), and the slack for making coke. The specific gravity is 1.272.

Analyses of Pratt Coal.

	I.	II.	III.	IV.
Authority.	Phillips.	McCalley.	Min. Resources of U. S., 1892.	Lupton.
Fixed carbon,	67.90	61.600	64.30	63.82
Volatile material,	29.80	31.480	32.08	31.85
Moisture,	......	1.508	1.07	1.02
Ash,	2.30	5.416	2.08	3.31
Sulphur,	0.83	0.918	0.47	0.70

Ultimate Analyses.

Authority.	I. Phillips.	II. Phillips.
Carbon,	75.82	75.05
Hydrogen,	10.52	9.91
Oxygen (by difference),	7.51	8.95
Nitrogen,	1.73	1.62
Sulphur,	1.07	0.97
Ash,	2.00	2.35
Moisture,	1.35	1.15
Total,	100.00	100.00

The coal from the mines is dumped on an ordinary bar-screen, with spaces 2½ inches in the clear; all going over this screen being shipped as lump. That which passes through is received on a shaking bar-screen, with ¾-inch spaces, which separates the nut from the slack. All the coal going through this screen is sent to the washer. Of an output of 700 to 800 tons per day, about 40 per cent. is shipped as lump and nut, and the remainder is washed for the coke-ovens.

The impurities occurring in the coal are pyrites, mineral charcoal, and slate partings. As delivered at the tip there will be also foreign slate (shale), and dirt from the top and bottom of the seam. The pyrites is found generally in thin sheets or local partings, and not in nodular form. The mineral charcoal also occurs in limited streaks; neither of these impurities forming regular partings of any extent. The slate parting is persistent, varying from a mere trace to a couple of inches in thickness, and occupying a constant position, about 8 inches from the roof. The other impurities mentioned are due entirely to careless mining. The pieces of slate and pyrites in the slack-coal are for the most part thin, and have a length and breadth several times as great as their thickness. The specific gravity of the slate is from 1.8 to 2.

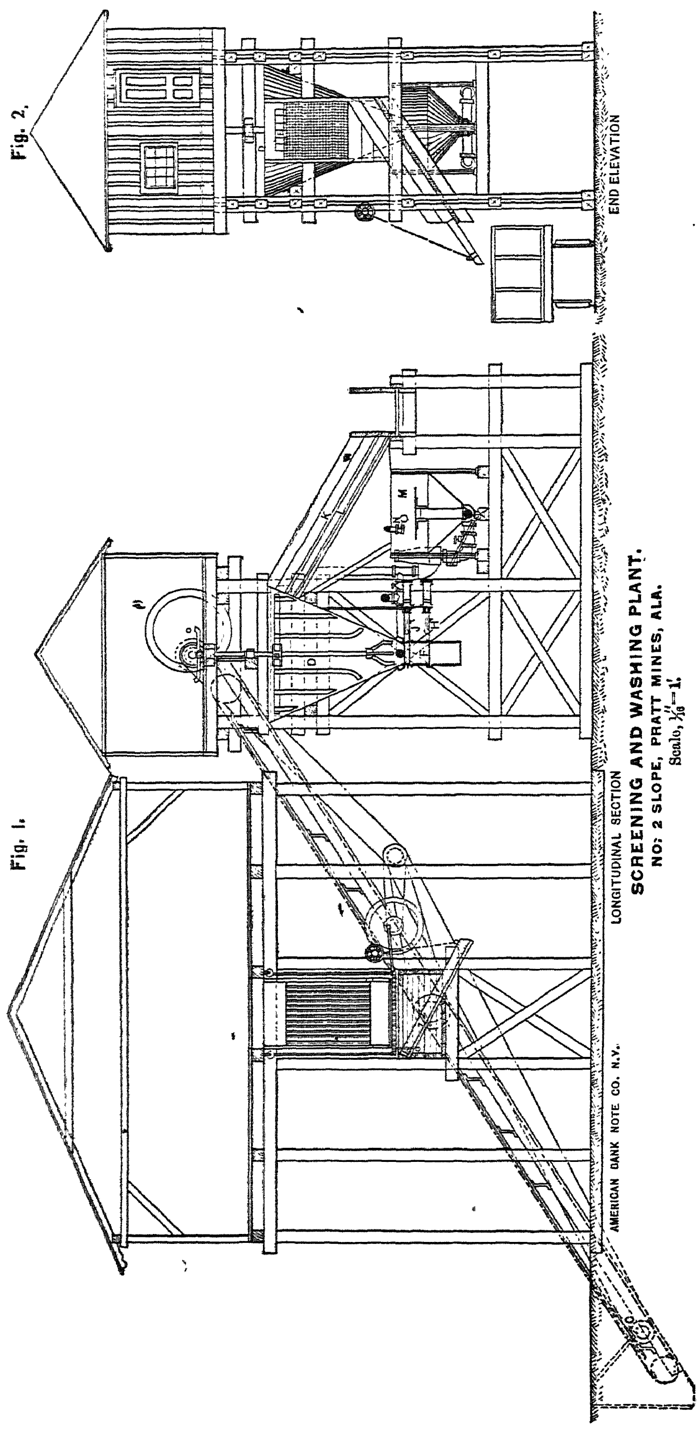
In mining the coal, single entries, with air-courses, are driven, and the workings are opened out by "room-and-pillar." The rooms are made 13 yards wide by 100 to 120 yards in length; the pillar left being 7 yards in breadth. All mining is done by hand, the coal is undercut with the pick and generally brought down by the use of black powder. Sometimes no explosives are needed. The bottom varies, being often a fire-clay, sometimes a soft, and again a very hard, slate. The roof

is a sandstone in some parts, a gray slate in others. Between the coal and the roof there is usually, but not always, a thin "muck" parting.

The Washing-Plant.

This consists of a 400-ton Robinson washer, with the necessary appliances for handling the coal before and after washing. The coal that passes through the nut-screen descends by gravity to a 16-inch screw-conveyor, with a pitch of 18 inches (A, Fig. 3). It is horizontal, 19 feet 6 inches long, and has, at a speed of 25 revolutions per minute, an actual capacity of 75 tons per hour. This screw delivers to a flight conveyor (B, Fig. 3) with a slope of 32 degrees, the flights being $7\frac{1}{2}$ by 13 inches and set 21 inches apart. As shown in the figures, the lower end of this conveyor is below the railroad-level, that it may take coal from the screw (C, Fig. 1), which is used at night, when coal from other mines is brought in by rail. The coal is delivered by this elevator over the central part of the washer-tub (D, Fig. 1). This is a cone-shaped tub of iron, 11 feet high, 11 feet 6 inches in diameter at the top and 22 inches at the bottom, the shell being $\frac{3}{8}$ inch in thickness. At the lower end is an annular compartment, connecting with the water-supply, and so perforated as to admit the water to the cone in the form of a number of small upward jets. In the center of the cone is a vertical shaft, reaching nearly to the bottom and carrying four wooden arms, to which are attached iron stirrers. Short stirrers are also attached directly to this shaft near its lower end. Motion is derived by means of gearing from an engine above.

The slack dropped from the conveyor into the washer starts to descend, but is met by the ascending currents of water, and the particles of coal are stopped in their downward career and carried up and over the discharge (E, Fig. 2), while the heavier impurities continue to the bottom. This separation is assisted by the continual agitation caused by the stirrers, which make 8 revolutions per minute, and are so arranged that the two sets travel different paths. The refuse material collects in the chamber (F, Fig. 1), closed at the bottom by the valve (H). When the attendant is satisfied that this chamber is full of slate, the valve (J) is to be closed and the lower valve (H) opened, discharging the waste into a car without at all interfering with the



SCREENING AND WASHING PLANT.
NO. 2 SLOPE, PRATT MINES, ALA.
Scale, 1/16"=1'.

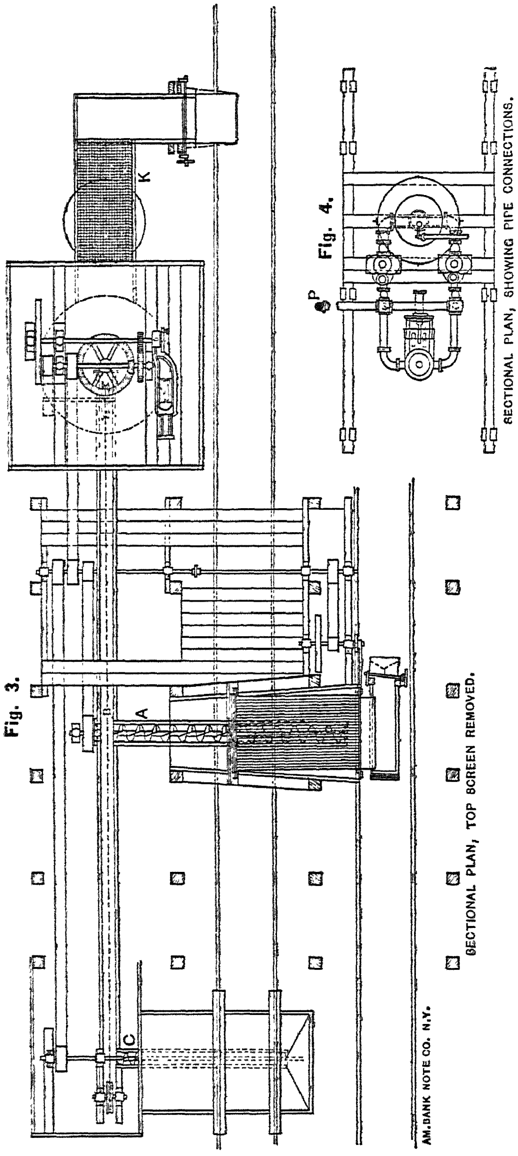
process of washing. But in practice the waste is allowed to accumulate in the bottom of the cone, and is emptied three or four times an hour by working the valves until it is certain that about all the refuse has been taken out. At first the valve-levers were operated by hand, requiring two and sometimes three stout men. But this method has been replaced by an arrangement of steam-pistons, so that the valves are now worked by one man without exertion. At the time when these notes were taken the slate was hauled away by a mule and driver; but it is intended to do away with this arrangement, and run the car by rope, so that one man can do all the work for the washer.

The cleaned coal and water passing the overflow (E, Fig. 2) are received on the screen (K, Figs. 1 and 3). At first there was but one screen, of steel, with $\frac{1}{8}$-inch perforations. It did not drain the coal satisfactorily, and wore out in a very short time. The present arrangement consists of two screens, both of manganese bronze. The upper one is $\frac{1}{4}$ inch thick, with $\frac{3}{8}$-inch perforations, $\frac{3}{4}$ inch from center to center. The inclination is 30 degrees, and the screen is $4\frac{1}{2}$ feet wide by 15 feet long, the last three feet, however, being blank. The fine coal and water that pass through this upper screen fall on the screen (L, Fig. 1), of No. 20 metal, having $\frac{1}{12}$-inch perforations; the coal from both screens discharging into a chute, which empties into the railroad cars. The water and sludge passing through the lower screen go to the tank (M, Fig. 1), from which the pulsometers draw.

In the English and the earlier American plants this tank was merely a "sump" for the pulsometers. But even with $\frac{1}{12}$-inch perforations there is a considerable amount of solid material—fine coal, slate and pyrites—contained in the water. As all the water, except that carried away by the washed coal, is used over again, the effects of the attrition of this material in the pumps and pipes is serious. Valves quickly wear out, and at one plant in the Birmingham district a pulsometer lasted only eighteen months. Again, with the simple tank this fine sediment—and especially the slate and pyrites—settles on the bottom, accumulating until it acquires a considerable height above the level of the discharge-pipe from the tank to the pumps. This, after a while, slips down with a rush and clogs up the

pumps to such an extent as to prevent them from working. Daily shovelling was required to overcome this annoyance.

After experiences of this sort at the Shaft No. 1 washer, Mr.



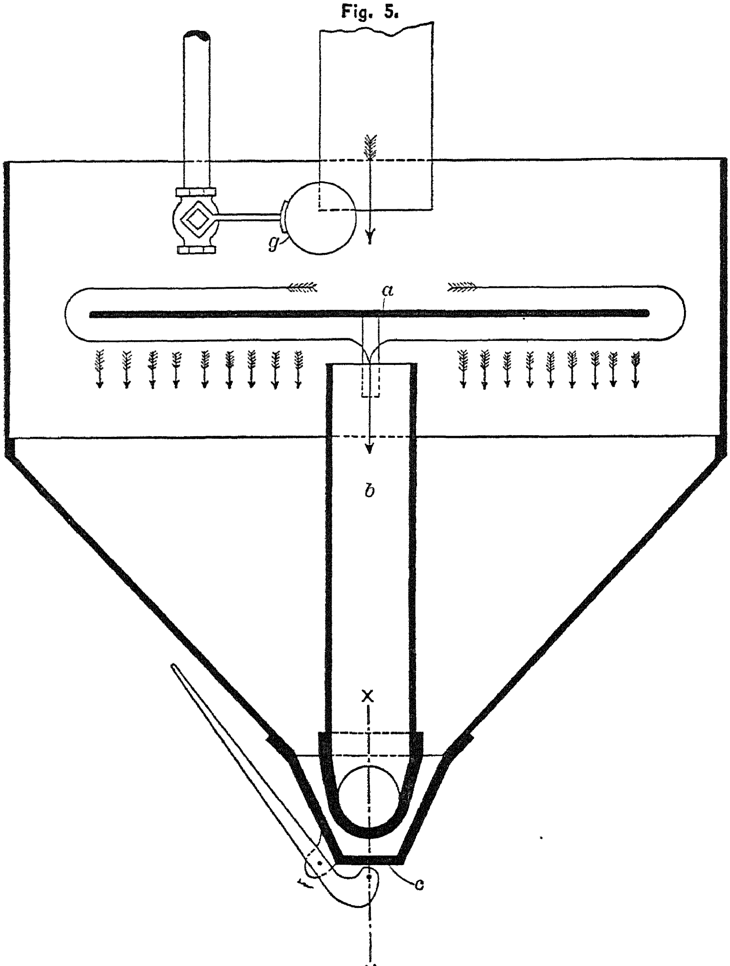
SCREENING AND WASHING PLANT,
NO. 2 SLOPE, PRATT MINES, ALA.
Scale, 1/16″ = 1′

Erskine Ramsey, Chief Engineer of the Tennessee Coal, Iron and Railroad Company, devised a tank that has been used at the No. 2 Slope plant with gratifying success. As shown in

Figs. 5 and 6, it is an iron tank, cylindrical in section at the top, funnel-shaped at the bottom. In this tank is a circular deflecting-plate (*a*, Fig. 5). The water, charged with fine coal and impurities, is delivered into the top and at the center, so that there may be an even distribution over the entire surface of the plate. The flow of the water, on entering the tank, is indicated by the arrows in Fig. 5. With this current of water are carried the fine coal-particles, while the impurities, owing to their greater specific gravity, drop from the current, as indicated in the sketch, into the comparatively still water below the level of the mouth of the pump-supply pipe (*b*, Fig. 5), and collect in the bottom of the tank. From here this refuse is removed by means of a valve (*c*) discharging the sludge into a trough, by which it is carried to the waste-car under the washer-tub. The relation between the diameters of the deflecting-plate and the tank is a point depending on the amounts of coal and of impurities in the fines and on the difference in specific gravity of these materials. With too small a plate the impurities will go to the pumps with the coal. With too large a diameter the coal will not be carried along with the current, but will be lost with the slate. Once regulated for a given coal, the results are distinctly good, as will be seen from analyses of refuse at the No. 2 Slope washer, given below. In connection with this tank is the valve for supplying the fresh water needed by the washer, automatically regulated by a float (*g*, Fig. 5, and N, Fig. 1).

The water, freed from its heavier impurities and augmented by the necessary amount from the fresh-supply pipe, is taken by the pulsometers through the central pipe (*b*, Fig. 5), and the connections (*e*, *e*, Fig. 6), and pumped directly into the washer-tub. This is an innovation on former practice, the old plan being to pump into a tank 40 to 60 feet above the bottom of the washer, with a discharge-pipe from this tank to the washer, in order to maintain a constant head. At this plant the same object is accomplished at less expense. The pipes between the pulsometers and the washer are connected to a stand-pipe (P, Fig. 4) 80 feet in height and open at the top. This acts as a balance on the inflowing current, and is of especial advantage when, as sometimes happens after a stoppage, the material in the washer becomes packed. The pumps then force water up

Fig. 5.



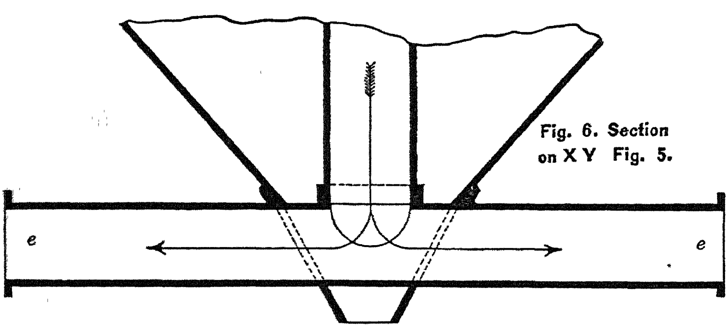


Fig. 6. Section on X Y Fig. 5.

SKETCH OF ERSKINE RAMSEY SLUDGE-TANK.

the stand-pipe, until a head is developed sufficient to force a way through the obstructing stuff. Seldom has this column-pipe overflowed.

The engine that drives the washer-machinery is single, 10 by 16 inches, with 3-inch steam-supply. It furnishes also the power for operating the two screws, the elevator, and the shaking screen. The steam-plant includes six boilers, each 46 inches in diameter by 26 feet long, with two 15-inch flues, and fired with "run-of-mines" coal. Three boilers are in use, carrying 85 to 90 pounds steam-pressure, and supplying steam for the pair of hoisting-engines at the slope as well as for the washer-engine. One fireman is employed.

One man does all the work at the washer. He must watch the engine and keep it and the other machinery oiled; operate the main slate-valves three or four times an hour, and also the sludge-tank valve, and load the washed coal into the railroad-cars. He is by no means overworked in attending to these duties, and will have ample time to run the refuse-car when the rope-haul for it is introduced. For the same capacity, even the trough-washers can hardly excel, if they can equal, this labor-record.

The cost of a Robinson washing-plant must vary with the particular conditions at each locality. Basing the estimate on the records of several plants in Alabama and Tennessee, the total cost of a 400-ton plant complete and ready for washing, including machinery for supplying the coal and disposing of it after washing, and also the royalty to the owners of the patent-rights, may be put at from $5000 to $8000. The cost of the washer-tub and its immediate appliances would be about $1000. The cost of repairs is low; in fact, to the washer proper, there will be almost no repairs needed. But water-valves, pumps, screens, elevators, etc., need attention and renewal from time to time, which are chargeable to the account of the washer.

Results.

Perhaps the first question arising is that of actual working capacity. At this Pratt mines plant the average output has been for many months fully up to the nominal capacity of 400 tons. Occasionally, for several hours at a time, the output has been at the rate of 600 and more tons, per day of ten hours.

It is not likely that the quality of the product on these occasions could have been equal to that obtained in treating a normal quantity. From its appearance to the eye this was indeed claimed; but no analyses were made to substantiate it. It may be noted here that the output in clean washed coal may be double the nominal capacity, when nut-coal free from slack is used. On the other hand, if only very fine material be used, for instance, coal from a disintegrator, probably not over 200 tons a day could be cleaned.

From the analyses of Pratt coal above quoted it will be seen that the average ash is about 3 per cent. These figures were obtained presumably from lump-coal. Table I. gives a series of analyses of the slack used at No. 2 Slope, taken during regular working of the plant and sampled between the last screen and the washer. The spaces between the screen-bars are ¾ inch in the clear; and everything that passes through this screen goes to the washer without further treatment.*

TABLE I.—*Slack Coal, Before Washing.*

Sampled 1893.	Volatile and Combustible Material.	Fixed Carbon.	Ash.	Sulphur.
November 1	30.53	63.28	6.19	1.53
" 2	27.64	60.20	12.16	1.50
" 3	30.12	61.86	8.02	1.55
" 4	29.11	60.41	10.48	1.57
" 6	29.15	59.01	11.84	1.27
" 7	30.41	63.51	6.08	1.65
" 8	29.28	63.42	7.30	1.49
" 9	30.45	58.25	10.30	1.39
" 10	30.23	57.03	12.74	1.60
" 11	28.64	56.66	14.70	1.34
Average	29.55	60.36	9.98	1.48

Table II. shows the results of investigations of the washed product, samples being taken from the coal as it was delivered to the railroad-cars.

It will be seen that the average ash in the coal has been reduced from 9.98 to 5.78 per cent. In other words, the washed coal contains 42 per cent. less ash than the unwashed. The

* These analyses and those in the other tables following were made by Dr. W. B. Phillips.

TABLE II.—*Washed Coal.*

Sampled 1893.	Volatile and Combustible Material.	Fixed Carbon.	Ash.	Sulphur.
November 1	25.63	68.69	5.68	1.42
" 2	29.76	63.65	6.59	1.50
" 3	28.66	66.43	4.91	1.46
" 4	30.92	61.55	7.53	1.19
" 6	31.00	61.37	7.63	1.31
" 7	31.10	63.93	4.90	0.86
" 8	31.64	63.78	4.58	1.12
" 9	32.59	64.05	3.36	1.17
" 10	32.61	61.19	6.20	1.28
" 11	33.01	60.49	6.50	1.24
Average	30.69	63.51	5.78	1.25

reduction in sulphur is over 15 per cent., and the gains in volatile material and fixed carbon are about 4 and 5 per cent. respectively. Table III. gives in detail the effects of washing, calculated from the above tables as percentages on the figures for the unwashed coal.

TABLE III.—*Comparison Between Tables I. and II.*

	Volatile and Combustible Matter.		Fixed Carbon.		Ash.		Sulphur.	
	Increase.	Decrease.	Increase.	Decrease.	Increase.	Decrease.	Increase.	Decrease.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
1	....	16.05	8.55	....	....	8.24	....	7.19
2	7.67	....	5.73	....	....	45.80	....	......
3	....	4.85	7.39	....	....	38.77	....	5.81
4	6.22	....	1.88	....	....	28.15	....	24.20
6	6.35	....	4.00	....	....	35.56	3.15	......
7	2.27	....	0.66	....	....	19.41	....	47.88
8	8.05	....	0.56	....	....	37.26	....	24.83
9	7.03	....	10.00	....	....	67.38	....	15.82
10	7.87	....	7.30	....	....	51.33	....	20.00
11	15.26	....	6.75	....	....	55.78	....	7.46
Average	3.86	....	5.22	....	....	42.08	....	15.54

Table IV. gives analyses of the washed coals of larger dimensions only, samples being taken from that part of the product which goes over the screen with ⅜-inch perforations.

The average of these results, compared with those of Table I, shows a reduction in ash of over 48 per cent., a reduction in

TABLE IV.—*Washed Coal, Over $\frac{3}{8}$-Inch Screen.*

Sampled 1893.	Volatile and Combustible Material.	Fixed Carbon.	Ash.	Sulphur.
November 1	31.13	63.82	5.05	1.46
" 2	29.08	64.99	5.93	1.47
" 3	30.40	65.07	4.53	1.23
" 4	29.82	66.48	3.70	1.50
" 6	30.07	63.98	5.95	1.09
" 7	31.14	63.71	5.15	1.42
" 8	30.99	63.75	5.26	1.02
" 9	32.01	63.35	4.64	1.13
" 10	32.44	62.28	5.28	1.25
" 11	33.06	60.82	6.12	1.12
Average	31.01	63.82	5.16	1.27

sulphur of nearly 15 per cent., and gains in volatile material and fixed carbon of about 5 and 6 per cent. respectively. A detailed statement of the results of Table IV. compared with those of Table I. is given in Table V.

TABLE V.—*Comparison Between Tables I. and IV.*

	Volatile and Combustible Matter.		Fixed Carbon.		Ash.		Sulphur.	
	Increase.	Decrease.	Increase.	Decrease.	Increase.	Decrease.	Increase.	Decrease.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
1	1.96	....	0.85	....	....	18.42	....	4.57
2	5.21	....	7.95	....	....	51.23	....	2.00
3	0.93	....	5.19	....	....	43.51	....	20.65
4	2.44	....	10.05	....	....	64.69	....	4.46
6	3.15	....	8.42	....	....	49.75	....	14.18
7	2.40	....	0.31	....	....	15.29	....	13.94
8	5.84	....	0.52	....	....	27.94	....	31.54
9	5.13	....	8.76	....	....	54.95	....	18.70
10	7.31	....	9.21	....	....	58.55	....	21.88
11	15.43	....	7.34	....	....	58.37	....	16.42
Average	4.93	....	5.73	....	....	48.29	....	14.77

It must be remembered that this washer is treating at one operation all sizes of coal from $\frac{3}{4}$ inch in thickness down to fine dust. Many pieces of the thickness named exceed it in their other dimensions, as is natural with a separation by bar-screen only. It could not be expected that a current and speed suita-

ble for the larger dimensions would make as good a separation of the finer materials.

Table VI. gives a series of analyses of the washed coals that pass through the $\frac{3}{8}$-inch holes and over the screen with $\frac{1}{12}$-inch perforations.

TABLE VI.—*Washed Coal, Under $\frac{3}{8}$-Inch Screen.*

Sampled.	Volatile and Combustible Material.	Fixed Carbon.	Ash.	Sulphur.
November 1	28.15	64.05	7.80	1.62
" 2	29.15	66.36	4.49	1.42
" 3	29.30	60.82	9.88	1.53
" 4	29.71	60.44	9.85	1.60
" 6	30.21	63.34	6.45	1.31
" 7	29.88	62.74	7.38	1.24
" 8	30.47	62.61	6.92	1.25
" 9	28.02	58.93	13.05	1.25
" 10	30.12	58.33	11.55	1.52
" 11	30.37	59.84	7.79	1.26
Average	29.54	61.75	8.52	1.40

Comparing the average results of Tables VI. and I. it is seen that the reduction in ash is about 14.5 per cent., in sulphur 6 per cent., with practically no change in volatile matter, and 2 per cent. increase in fixed carbon. A detailed comparison is given in Table VII.

TABLE VII.—*Comparison Between Tables I. and VI.*

	Volatile and Combustible Material.		Fixed Carbon.		Ash.		Sulphur.	
	Increase.	Decrease.	Increase.	Decrease.	Increase.	Decrease.	Increase.	Decrease.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
1		7.80	1.21		26.01		5.88	
2	5.82		10.23			63.08		5.33
3		2.72		1.68	23.19			1.29
4	2.06		0.05			6.01	1.91	
6	3.63		7.34			45.52	3.15	
7		1.74		1.21	21.38			24.85
8	4.07			1.27		5.21		16.10
9		7.98	1.16		26.70			10.07
10		0.36	2.28			9.34		5.00
11	6.04		5.61			47.01		5.97
Average		0.03	2.30			14.64		6.04

A glance at this table and at Table V. shows at once that the washer, working under existing conditions, is better adapted to the larger-sized coals than to the fines. Yet it is economical to continue as at present, since the amount of product passing through the $\frac{3}{8}$-inch screen is small, not over 10 per cent. of the total, and for this amount a secondary treatment would scarcely pay.

With regard to the composition of the refuse, two sets of analyses are presented, the first (Table VIII.) being of coarse material taken from the main washer-tub, the second (Table IX.) of the fine stuff from the sludge-tank.

TABLE VIII.—*Coarse Slate Refuse.*

Sampled.	Volatile and Combustible Material.	Fixed Carbon.	Ash.	Sulphur.	From Solution, Sp. gr. 1.235.	
					Coal.	Ash in Coal.
					Per cent.	Per cent.
November 1	14.04	19.85	66.11	2.91	11.00	37.20
" 2	14.46	31.14	54.40	2.11	6.81	13.20
" 3	19.27	34. 6	45.97	1.93	26.20	23.10
" 4	16.25	25.96	57.79	2.41	14.17	33.20
" 6	15.15	24.54	60.31	1.79	12.50	24.80
" 7	18.10	35.39	46.51	2.09	21.80	23.50
" 8	13.63	23.43	62.94	1.67	9.52	23.60
" 9	14.71	21.44	63.85	1.53	4.87	11.86
" 10	11.82	15.80	72.38	1.81	......	......
" 11	12.79	17.02	70.19	1.66	3.80	18.03
Average	15.02	24.93	60.05	1.99	12.29	23.16

As the amount of this refuse in a day's run is about 18 tons, and the coal-contents, as shown by the above table, are 12.29 per cent., there will be 2.3 tons of coal lost in the average run. The irregularities in coal contents are due partly to the use of a bar-screen and partly to the work of the attendant. Large lumps of coal are occasionally passed by the screen, and of course descend with the slate. The attendant may sometimes open the valves too often, and cause a loss of coal.

The value of Mr. Ramsey's tank in getting rid of the worthless material is shown by the following analyses. About 7 tons per day are drawn from it, only 16 per cent. of which, or 1.1 tons, is coal. The entire loss in coal then, on 425 tons of material treated, is 3.4 tons or 0.8 per cent. of the total. The

TABLE IX.—*Refuse from Ramsey Sludge-Tank.*

Sampled.	Volatile and Combustible Material.	Fixed Carbon.	Ash.	Sulphur.	From Solution, Sp. gr. 1.235.	
					Coal.	Ash in Coal.
					Per cent.	Per cent.
November 1	17.85	34.20	47.85	2.93	7.40	8.60
" 2	17.96	26.00	56.04	2.43	6.81	13.20
" 3	16.31	29.25	54.44	2.72	14.00	14.60
" 4	23.39	46.78	29.83	2.45	32.80	10.60
" 6	24.29	46.59	29.12	2.19	25.27	9.80
" 7	22.94	48.24	28.82	2.23	23.00	8.20
" 8	21.91	46.54	31.55	2.15	8.90	7.00
" 9	19.85	33.40	46.75	1.99	17.35	7.75
" 10	15.25	23.13	61.62	2.08	3.68	10.87
" 11	20.16	39.41	40.43	2.48	21.17	10.00
Average	19.99	37.35	42.65	2.36	16.03	10.06

total refuse material, slate and coal together, is 25 tons, or 6 per cent.

The amount of fresh water needed to take the place of that carried off with the refuse and washed coal was found to be 14,050 gallons. On the day of this test 400 tons of washed coal were produced, and the washer was running for 11 hours. The average water per ton of washed coal was 35.1 gallons; average per minute, 21.3 gallons. Hourly measurements were taken, showing from 24 to 51.2 gallons of water per ton of coal. This irregularity was due to the varying coal-supply which, depending on the way coal came out of the mine, was sometimes only 25 tons (washed) in an hour.

The cost of washing per ton of washed coal is low. The daily expenses may be estimated as follows:

For labor at washer,	\$2.00
For labor at boilers, fuel, etc.,	4.00
For repairs and supplies,	3.00
Total,	\$9.00

This for 400 tons would be 2.25 cents per ton; and it is quite likely that the actual figures are still lower.

THE COKE.

The washed coal is carried in hopper-bottomed railroad cars to the ovens, and there dumped into a series of bins of 5000

tons' capacity. From these it is loaded into 6-ton larries, hauled in trips of two by small steam locomotives. The ovens are all of the bee-hive pattern, 12 feet in diameter, and built with a height of 7 feet 6 inches, though the average height now is probably not over 6 feet 9 inches. The outside walls are of sandstone, the oven-walls of fire-brick. Of the bottoms, some are of fire-brick; some of 12 by 12 by 3-inch fire-brick tiles; some of common red brick. They were built with the back 6 inches higher than the front, but many have no slope now. With unwashed coal the usual charge was from 4 to 4.5 tons. Since using washed coal this has been increased to about 6 tons, without any increase of wages to the pullers, as the labor is less than when pulling coke made from unwashed coal. The ovens retain the heat better than before, in spite of the washed coal being charged damp. Repairs to ovens are less than before using washed coal. All coke is quenched in the ovens.

TABLE X.—48-*Hour Coke from Unwashed Coal.*

Sampled 1894.	Volatile and Combustible Material.	Fixed Carbon.	Ash.	Sulphur.
January 9	0.86	89.48	9.66	1.24
"	0.45	84.03	15.60	1.37
"	0.50	87.28	12.20	1.21
"	0.80	84.75	14.35	1.33
"	0.50	83.00	16.50	1.43
Average	0.62	85.70	13.66	1.31

TABLE XI.—48-*Hour Coke from Washed Coal.*

Sampled 1894.	Volatile and Combustible Material.	Fixed Carbon.	Ash.	Sulphur.
January 9	0.40	90.00	9.60	0.88
"	0.50	89.06	10.44	1.25
"	0.40	89.40	10.20	1.03
"	0.50	88.40	11.20	1.44
"	0.90	88.35	10.75	1.05
Average	0.54	89.04	10.43	1.13

A comparison of Tables X. and XI. shows that there was in the samples taken an increase of 3.9 per cent. in fixed carbon, a decrease of 23.6 per cent. in ash, and a decrease of 13.7 per

cent. in sulphur, due to washing. A week's record of washed coke samples from stock house shows:

	1.	2.	3.	4.	5.	6.	7.	Average.
Ash......................	10.68	8.80	9.57	10.40	9.90	9.40	9.40	9.73

Coke from the washed coal can be recognized at the door of the oven by the difference in the amount of braize. To determine the improvement in this respect, the weights of the ash-piles in front of a number of ovens were carefully taken, showing the average amount, when coke is made from unwashed coal, to be 521 pounds, and, with washed coal, 238 pounds, or a saving of 283 pounds of coke per oven. If the output from each oven is taken at 2.5 tons (the tests having been made with the same charge as customary when using unwashed coal) the saving is 5.66 per cent. There will also be saved a certain amount of the braize made in forking the coke from the oven-door to the car, in the unloading of the cars, and the loading into furnace-buggies. Weights at the furnaces of braize left in cars after unloading showed 3 per cent. in the case of unwashed coal, and 1 per cent. when washed coal had been used.

This gain in output of marketable coke is sufficient, without charging the furnaces any higher price for their fuel, to compensate the mines for the cost of washing, and for the material formerly put into the ovens but now sent to the waste dump. Assuming a selling-price of $2 per ton, the saving in braize at the ovens is 11.32 cents, in the cars 4 cents, or in both items 15.32 cents per ton of coke. The refuse from washer, formerly coked, is 6 per cent. of the total. To make a ton of coke, 1.67 tons of coal are required. Six per cent. of this, or 0.1 ton, may be called, from the standpoint of the mines, the loss in "coal" per ton of coke. Assuming, as an average cost of coal, 80 cents per ton, the increase in cost of coal per ton of coke is 8 cents. To this must be added the cost of washing, 2.25 cents per ton of coal, or, 3.75 cents per ton of coke. The total is 11.75 cents, against which there is, as above, a saving of 15.32 cents, or a net saving of 3.57 cents per ton of coke, due to washing.

In the furnace, the washed coke is distinctly advantageous.

There is less of that fine stuff from which no valuable service is realized. Comparative tests of crushing strength have not been made; but the "washed coke" undoubtedly will sustain a heavier burden than the unwashed. A few words from a letter of an official of one of the Birmingham companies will show the estimation in which the once despised washers are now held:

"The cost of coke per ton of iron made will be about 50 cents less for the month of March on the furnaces using washed coke. From the present work of the coke in the furnaces it would pay to wash the coal, even though all the waste was coal."

Practical operations in Alabama, Georgia, and Tennessee, during the past four or five years, have proved that this washer is well adapted to such coals as those of the southern field, containing a moderate quantity of impurities. Its advantages may be summed up as follows:

a. Low first cost.
b. Low labor-cost.
c. Compactness of plant.
d. Economy of water.
e. Small waste of coal.
f. Ability to treat with good results materials not closely sized.

Milling Arizona Gold-Ores with a "Colorado" Stamp-Mill.

BY WILLARD S. MORSE, PRESCOTT, ARIZONA.

(Florida Meeting, March, 1895.)

REFERRING to Mr. Rickard's paper on "The Limitations of the Gold Stamp-Mill" (*Trans.*, xxiii., 137), and the discussions that have followed, and without entering into any controversy as to the relative merits of the "California" and "Colorado" types of stamp-mills, I wish to give the results obtained on ores from Lynx Creek district, near Prescott, Arizona, with a stamp-mill of the Colorado, or, more precisely, the Gilpin county, Colo., type.

The mines of the district have been worked for nearly thirty

years, yet in that time very little, if any, work has been done on the veins below the line where the oxidized or "free" ores end, and the sulphide or "base" ores come in, except in a few cases where the sulphide-ore was high enough in value to ship to smelters. The surface or oxidized ores have been worked in arrastras and stamp-mills, but few attempts have been made to mill the so-called "base" ores. About thirteen years ago a smelter was built in the district by Mr. John Howell to smelt these ores, but was abandoned on account of the high transportation charges on fuel and bullion.

The saving shown in this paper is not claimed to be high, and the history of the district has been given to show that heretofore, at least, the ores have not been considered suitable for stamp-milling.

The ore from which the results are given was extracted from below water-line (100 to 250 feet from the surface), and is a quartz carrying zinc-blende, iron pyrites, galena, and a small percentage of copper and arsenical pyrites.

MILL.

The mill is a typical "Gilpin County" stamp-mill of 10 stamps. No rock-breaker or self-feeders are used, the ore being fed by hand. I do not wish to be understood as advocating this method of feeding. It was adopted as a matter of economy in the first cost of plant, as the attempt to mill these ores was regarded as an experiment, in view of the history of the district.

The weight of stamps when new was as follows:

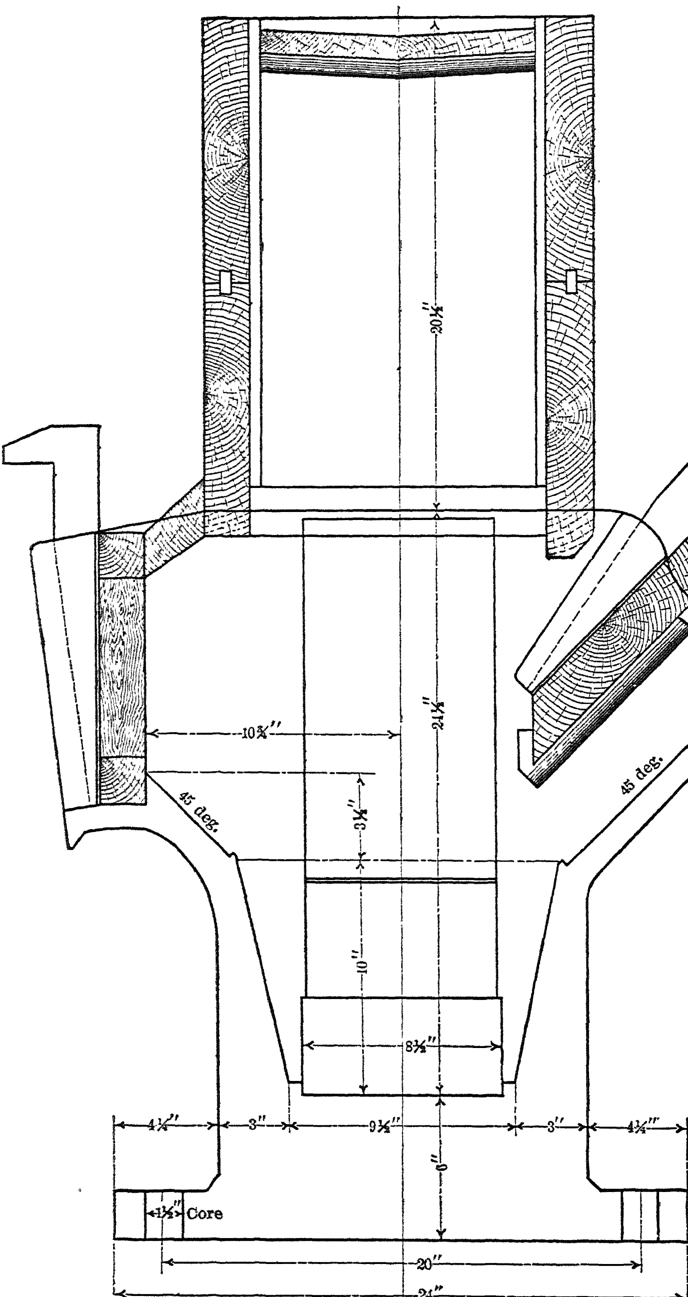
	Pounds.
Stem,	265
Tappet,	35
Head,	225
Shoe,	85
Total,	610

The stamps dropped 15 inches, 36 times per minute, in the following order: 1–5–2–4–3.

Fig. 1 shows a section of the mortar.

The mortars are provided with copper amalgamating-plates, the front plate being 5 inches, and the back plate 10 inches, wide. Both plates extend the full length of the mortar, and have an inclination of 45°.

FIG. 1.



Section of Mortar.

The outside plates (one for each battery of 5 stamps) are 96 by 52 inches, silver-plated (1 ounce to the square foot). These plates are set with an inclination of 1⅝ inches to the foot.

Concentrators.

For concentration of the tailings after amalgamation, two Gilpin county bumping or percussion tables are used. The beds of these tables are made of cast-iron. The cam-shaft of the table is run at 78 revolutions per minute, giving the table 156 strokes or bumps per minute.

Calculations.

The calculations presented are based on the following data:

Tailings.—A sample of tailings running from the mill is taken every half-hour by diverting the entire stream of tailings through a swinging trough, which discharges into a galvanized iron tub. This trough is operated by a cord from the battery. The sample thus collected, containing the proper proportion of slimes and sands, is decanted after completely settling and evaporated to dryness. Two such samples are made daily and assayed, and the results given below are the average of 503 samples and assays.

Concentrates.—The weights, assays, and analyses of concentrates given are from smelter-returns.

Bullion.—United States mint returns are used for contents of bullion.

Ore.—The assay-value of ore has been determined by calculation based on the weight of ore and concentrates and the contents of bullion, concentrates, and tailings.

Results.

The following are the results obtained from the milling of 2432.9 tons of ore of an average assay-value of 0.763 ounce of gold per ton, which varied in monthly runs from 0.574 to 1.18 ounces per ton:

Amalgam.—Total amalgam recovered, 5711.6 ounces. Of this, 70.2 per cent. was from inside battery plates, and 29.8 per cent. from outside plates.

Retort.—Weight of retort, 2024.45 ounces, or 35.4 per cent. of weight of amalgam.

Bullion.—Weight of bar, 1854.38 ounces. Loss of weight in

melting retort, 8.4 per cent. Assay of bullion, gold, .636 fine. Contents of bullion, gold, 1180.148 ounces.

Concentrates.—Net weight of concentrates, 605,149 pounds avoirdupois, or 12.4 per cent. of weight of ore. Assay and analysis of concentrates, Au, 1.347 ounces per ton; Ag, 6.93 ounces per ton; Pb, 6.34 per cent.; SiO_2, 9.9 per cent.; Fe, 30 per cent.; Zn, 6.85 per cent. Contents of concentrates, 407.6972 ounces of gold.

Tailings.—Weight of ore, 4,865,822 pounds; weight of concentrates, 605,149 pounds; weight of tailings, 4,260,673 pounds. Assay of tailings: Average of 503 samples and assays, 0.1271 ounces of gold per ton. Contents of tailings, 270.8327 ounces of gold.

Calculation of Saving and Loss.

	Ounces Gold.	Per cent.
Bullion	1180.1480	63.5
Concentrates	407.6972	21.9
Tailings	270.8327	14.6
	1858.6779	100.

The highest result obtained by amalgamation was on a lot of 332 tons, assay-value of ore 1.134 ounces of gold per ton, which was:

	Per cent.
By amalgamation	76.3
In concentrates	14.1
Lost in tailings	9.6

Sizing of Tailings and Concentrates.

Mesh of Screen used on Battery.	Material.	Coarser than 60-mesh.	Through 60-mesh and remaining on 80-mesh.	Through 80-mesh and remaining on 100-mesh.	Through 100-mesh and remaining on 150-mesh.	Through 150-mesh and remaining on 200-mesh.	Finer than 200-mesh.
		Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
30	Tailings	5.4	3.9	15.6	12.3	11.0	51.8
30	Concentrates	2.8	2.4	7.8	16.	19.4	51.6
40	Tailings	6.6	5.	10.8	13.6	6.8	57.2
40	Concentrates	1.9	1.1	5.9	16.5	22.1	52.5
60	Tailings	....	2.	3.	14.5	9.5	66.
60	Concentrates	....	2.	3.4	10.4	20.	58.2

Tests made by panning weighed quantities of tailings and weighing and assaying the concentrates thus recovered, showed that about 60 per cent. of the gold lost in tailings could be accounted for in this way. The concentrates thus saved, however, invariably assayed much lower than the average of the concentrates that were saved on the concentrating-tables, and were very fine, 90 per cent. passing through a 200-mesh screen.

Sized samples of concentrates, each size assayed separately, invariably show that the finer concentrates assay less than the coarser sizes.

The following results on a sample of concentrates assaying 1.8 ounces of gold per ton will serve as an illustration of one of many such experiments that have been made:

Size and Assay of Concentrates.

Size.	Assay. Ounces gold per ton.
Coarser than 60-mesh	2.6
Through 60-mesh, remaining on 100-mesh	2.2
Through 100-mesh, remaining on 150-mesh	2.2
Through 150-mesh, remaining on 200-mesh	2.
Finer than 200-mesh	1.5

Conclusions.

From more than 80 assays made on specimens of the various characters of ore found in the district, the following conclusions have been arrived at:

1. That the gold is contained in the quartz and "free," or else is associated with the iron, copper, or arsenical pyrites, and that a large percentage of that associated with the pyrites can be extracted by amalgamation with very fine crushing.

2. That the zinc-blende, as a rule, carries very little gold-value, and that in cases where any considerable quantity of gold has been found in the blende, it was very "free," and easily extracted by amalgamation after fine crushing.

3. That the galena, as a rule, assays low in gold and high in silver.

Notes.

Screens.—40-mesh burr-slot screens are used as a rule, and last about three weeks.

Shoes and Dies.—Cast-iron shoes and dies are used, and the actual wear of iron per ton of ore is: for shoes, 1.122 pounds; for dies, 0.692 pound.

Crushing-Rate.—The average crushing-rate of the mill for nine months, based on actual running time, is 3355 pounds per stamp per twenty-four hours.

Water Used in Battery.—Water used in the battery, 21,000 gallons per twenty-four hours, or 1252 gallons per ton of ore crushed.

Results on Ore High in Zinc.

The results on a small lot of ore selected for high zinc-contents may be of interest.

The assay and analysis of the ore are calculated from contents and analysis of concentrates and tailings, weight of ore, and concentrates and contents of bullion.

Ore.—Assay-value of ore, 1.393 ounces of gold per ton. Analysis, Pb, 2.90; SiO_2, 59.0; Fe, 8.8; Zn, 9.7 per cent.

Amalgamation.—The ore yielded by amalgamation 0.9 ounce fine gold per ton.

Concentrates.—One ton of concentrates was made to 3.8 tons of ore. Assay and analysis of concentrates: Gold, 1.54 ounces; Pb, 7.4; SiO_2, 9.8; Fe, 21.6; Zn, 19 per cent.

Tailings.—Assay and analysis of tailings: Gold, 0.12 ounce; Pb, 1.3; SiO_2, 76.6; Fe, 4.3; Zn, 6.5 per cent.

From the above data the following calculation is made:

	Contained in Bullion.	Contained in Concentrates.	Contained in Tailings.
	Per cent.	Per cent.	Per cent.
Gold	64.5	29.1	6.4
Lead	....	67.	33.
Zinc.	....	51.	49.
Iron	....	64.2	35.8
Silica	....	4.4	95.6

The Lixiviation of Silver-Ores by the Russell Process at Aspen, Colorado.

BY WILLARD S. MORSE, PRESCOTT, ARIZONA.

(Florida Meeting, March, 1895.)

THE purpose of this paper is to record the results obtained in the use of the Russell process at Aspen, Colo., covering a period of fourteen months, from November, 1891, to December, 1892, during which time over 30,000 tons of ore, purchased outright, on sample and assay, were treated. No explanation of results will be attempted, nor will the chemistry of the process be discussed.

The plant was designed by C. A. Stetefeldt, of San Francisco, and built by the owners by day-labor. The entire plant is run by water-power, supplied by a flume 3600 feet long, and a pipe-line 2200 feet long, with a head of 170 feet. A Pelton water-wheel, 8 feet in diameter, is used, from which power is transmitted by wire-rope to the main counter-shaft in the mill, a distance of about 250 feet. From the mill to the sampling-works, about 150 feet, power is also transmitted by a wire-rope.

A Pelton water-wheel, 3 feet in diameter, supplies power to the dynamo for lighting the plant.

The works are located adjoining the Santa Fe railroad tracks, and all ore and supplies are unloaded directly from the cars to the works.

The sampling-works are provided with crushers and rolls, the latter being used only on samples, and not on the entire ore. The Cornish hand-quartering method is used in sampling ore for purchase. After crushing and sampling, the ore goes to the bedding-floor, where it is made up into mixtures suitable for the mill.

The drying-plant consists of four Stetefeldt double shelf-dryers, six shelves high, and is provided with dust-chambers and a stack 65 feet high for draft. These dryers are fired with gas

from Taylor gas-producers, a description of which, together with the results obtained, was given in a paper by the writer, read at the Montreal Meeting, February, 1893,* to which reference is made for details omitted in this paper.

The crushing-plant consists of thirty 850-pound stamps, dropping ninety-two times per minute, with double-discharge mortars for crushing ore, and ten 650-pound stamps for crushing salt.

The chloridizing roasting is done in a Stetefeldt furnace, provided with very complete dust-chambers, and a stack 165 feet high for draft. The furnace is likewise fired with producer-gas, and the results will be found in the paper referred to above.

A cooling-floor, 100 by 125 feet, is used for cooling the ore after roasting.

The leaching-department is provided with seven leaching- or ore-tanks, 17 feet inside diameter and 9 feet deep; six precipitating-tanks for solution, 12 feet in diameter and 9 feet deep; six precipitating-tanks for wash-water, 8 feet in diameter and 10 feet deep; four storage-tanks for solution, 12 feet in diameter and 8 feet deep; and two tanks for the storage of sulphides, 10 feet in diameter and 3 feet deep.

A Johnson filter-press, operated by compressed air, and a steam-dryer, are used for pressing and drying sulphides.

Two Knowles air-compressors furnish compressed air for elevating solutions, stirring solutions during precipitation, and pressing sulphides.

Two boilers, 54 inches by 14 feet, supply steam for heating the mill and solutions, and for drying sulphides.

The following is an explanation of the terms used in this paper:

"Battery-sample"—the sample of ore after crushing, taken every half hour and assayed daily.

"Top-sample"—the sample of ore and salt mixed, taken every half hour at the top of the furnace, and assayed daily and solubility determined.

"Furnace-sample"—the samples of roasted ore taken as the ore is drawn from the furnace, and taken from the shaft of the furnace, the return-flue, and the dust-chambers.

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* *Trans.*, xxi., 919.

"Charge-sample"—the sample of roasted ore taken when the ore is charged to the leaching-tanks.

"Washed-ore sample"—the sample taken from the leaching-tanks after the ore has been leached with water and the soluble salts have been removed.

"Chlorination"—the amount or percentage of silver that can be extracted in the laboratory by leaching with hyposulphite of soda.

"Extraction"—the amount or percentage of silver that can be extracted in the laboratory by the Russell process.

"Solubility"—the percentage of soluble salts that can be removed in the laboratory by leaching with water.

"Ordinary solution"—a solution of about 2 per cent. of hyposulphite of soda.

"Extra-solution"—the cuprous hyposulphite solution of the Russell process.

Ores Treated.

The ores treated were principally from the Aspen district, and the following is the average composition, calculated from analyses made on each of nearly one thousand lots of ore:

	Ounces per ton.
Silver,	27.918
	Per cent.
Lead,	2.277
Silica,	21.663
Barite,	20.924
Lime,	10.992
Magnesia,	4.245
Iron,	10.025
Zinc,	2.854
Copper,	0.161
Sulphur,	8.105

The extremes of each are given below, from the analyses on lots of ore of about 30 tons each. The maximum for each is printed in heavy type, and the other figures in the same horizontal lines show the percentage of the other ingredients accompanying the said maximum.

The total ore treated during the period was 30,856.9445 tons (dry), assaying 27.9188 ounces of silver per ton and containing

861,488.05 ounces of silver. The value was ascertained by sampling and assaying, and was the amount paid for. It was checked by battery weights and daily samples, which showed an average of 27.9806 ounces of silver per ton; also by battery-weights and calculated value of top-samples made daily, which showed an average of 27.9 ounces of silver per ton.

Extreme of—	Pb.	SiO_2.	$BaSO_4$.	CaO.	MgO.	Fe.	Zn.	Cu.	S, as Sulphide.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Lead	**10.5**	15.0	35.0	6.6	1.2	13.2	1.8	....	0.3
Silica	....	**90.4**	....	....	....	2.5	2.2	....	3.0
Barite	1.1	3.8	**76.4**	5.6	1.6	1.3	1.3	....	....
Lime	5.0	3.0	....	**40.0**	7.5	2.2	1.0	....	....
Magnesia	1.3	12.0	4.0	24.3	**15.0**	3.2	1.8	....	....
Iron	0.5	3.0	....	....	....	**44.6**	2.9	....	45.1
Zinc	4.0	7.6	....	....	....	26.8	**15.8**	....	31.3
Copper	2.5	15.0	....	....	....	34.9	5 2	**4.7**	36.9

Of the 30,857 tons treated, 19,191 tons were sampled by the Taylor & Brunton sampling-works, using Brunton's patent automatic sampler, which "checked" with the mill-sampling by hand-quartering within 0.12 ounce of silver per ton.

Dust-Loss and Loss by Volatilization in Roasting.

	Tons.	Silver Assay. Ounces per ton.	Total Contents of Silver, Ounces.
Raw ore, ...	30,857	27.9188	861,488.05
Roasted ore, ...	31,775	24.6289	782,586.09

Loss of silver in ounces, ... 78,901.96
Or 9.157 per cent. of the total contents.

The above loss includes mechanical or dust-loss in sampling, drying, crushing, roasting, handling on cooling-floor and charging to leaching-vats. Just what proportion of this loss is chargeable to these causes it is hard to determine, but I question if it exceeds 1 per cent. The mill is provided with every appliance for preventing loss from these sources. The dust-loss in drying is reduced to a minimum by the use of shelf-dryers; the batteries are provided with a suction-fan, which forces the dust into a settling-chamber, and the dust-chambers of the Stetefeldt furnace seem to do the work for which they were

designed, as the last compartment contained more condensed fume than dust. It is possible that I underestimate the dust-loss, but I think not, when the fact is taken into consideration that samples of dust, taken at various times from the timbers of the mill, never exceeded two-thirds of the assay-value of the ore, and the dust caught in the dust-chambers of the Stetefeldt furnace showed an average assay-value of 18 ounces, against an average of 27 ounces of silver per ton from the shaft of the furnace.

I presume my figures will be criticized by Mr. Stetefeldt, who, I believe, has always claimed that the loss of silver by volatilization in roasting in the Stetefeldt furnace must necessarily be low, on account of the short exposure of the ore to a high heat; but I have never seen any figures from actual experience to prove this assertion.

Great care was taken to find out exactly what this loss was. For about three-quarters of the time every pound of the roasted ore was weighed while being charged to the leaching-tanks, and during the rest of the time the weight of the roasted ore was estimated by volume, based on the experience gained by actual weighing. The results checked closely with the period when the ore was actually weighed. A further confirmation of the correctness of the figures given is found in the results of a later run of about 8000 tons of ore, where all the roasted ore was weighed, and the loss showed over 10 per cent.

Roasting and Chloridizing.

The ore was roasted with 12.2 per cent. of its weight of salt, or 244.4 pounds per ton of ore.

Furnace-Samples.—Two daily determinations were made by taking a sample from each car of roasted ore as drawn from the shaft, return-flue and dust-chambers of the Stetefeldt furnace. The proportion of ore taken from each was noted, and assay-, solubility-, chlorination- and extraction-tests were made on each sample. Again, when the ore was charged into the leaching-vats another sample was taken, and the same determinations were made, and the increase in chlorination and extraction by lying on the cooling-floor was calculated.

From the data thus obtained the following averages have been calculated:

	Results When Drawn from Furnace.				
	Proportion of entire ore.	Ounces silver per ton.	Solubility.	Chlorination.	Extraction.
	Per cent.		Per cent.	Per cent.	Per cent.
Shaft of furnace	53.1314	27.7783	15.4524	43.5355	70.4077
Return-flue	33.0409	25.0434	10.3801	59.1006	83.8121
Dust-chambers	13.8247	18.3307	17.2105	64.1864	69.8810
General average	100.	25.4478	13.5072	52.5102	78.4358

After lying on the cooling-floor for an average of 102 hours, the ore showed the following results:

Ounces silver per ton.	Solubility.	Chlorination.	Extraction.
	Per cent.	Per cent.	Per cent.
24.6289	14.0830	78.9282	89.7857

Difference in results by lying on cooling-floor:

	Per cent.
Gain in chlorination, ...	26.4180
Gain in extraction, ...	11.3499

Leaching of Ore and Extraction in Mill.

The weight of the roasted ore treated was 31,775.338 tons, which was leached in 546 charges, averaging 58.19 tons. Each charge required about 22.5 cubic feet of water, or about 60 inches in depth in the tank, to completely saturate it. After washing and removing soluble salts, about 50 inches of solution will saturate the charge.

The following method of leaching was followed, as a rule:

1. The ore was charged into one volume of water, followed by a second volume of wash-water, and by:

2. Three volumes of warm ordinary solution (about 1.8 per cent. of hyposulphite);

3. One volume of extra-solution (average about 0.5 per cent. of $CuSO_4$);

4. One volume of warm ordinary solution;

5. One volume of extra-solution;

6. Two volumes of warm ordinary solution; and

7. The last wash-water, the dividing line between solution and wash-water being drawn when about one and one-fourth volumes of wash-water had been applied.

This treatment was varied as to quantity of solution applied and strength of extra-solution, but the above is about an average. The average time of leaching each charge was about 100 hours. This includes stops of every kind.

The average leaching-rate was 13 inches per hour. This was the natural leaching-rate; but, as a rule, in leaching after the ore had been washed, the leaching-rate was cut down to 10 inches per hour.

The following method of sampling the ore in the vats was used for all samples except the final tailings-sample. A brass tube with a valve in the bottom was used, and three to five cores one and one-fourth inches in diameter for the entire depth of the charge were secured. For the final tailings-sample the following method was used: after one-half of the charge had been sluiced, and a face of tailings about seven feet high and seventeen feet across thus secured, three samples were taken as follows: top-sample, taken about one foot from surface of ore and across entire face; bottom-sample, taken in same way about one foot from the bottom of the vat; and general sample, taken over the entire face and representing a general average of the tailings.

In addition to the intermediate samples, the following samples were always taken: washed-ore sample, taken after washing ore with water, on which chlorination- and extraction-tests were made, as well as assay; first ordinary sample, taken after the ore had been leached with ordinary solution, and on which the extraction in mill by hyposulphite is calculated; preliminary sample, taken before sluicing the tailings; and the *top*, *bottom*, and *general tailings-samples*, already described.

On the general tailings-samples, the contents of the tailings are calculated.

The average of these samples for the run are as follows:

	Ounces Silver per ton.
Washed-ore samples,	24.4905
First ordinary,	11.8174
Preliminary tailings,	3.44
Top-tailings,	4.04
Bottom-tailings,	4.86
General tailings,	3.7984

The following are the assays and laboratory-determinations made on samples, and the calculations based thereon:

Charge-samples,	Silver per ton, ...	24.6289 ounces.
	Soluble salts, ...	14.083 per cent.
	Calculated value (after deducting soluble salts), .	28.6648 ounces.
	Chlorination, ...	78.9282 per cent.
	Extraction, ...	89.7857 "
Washed-ore samples,	Silver per ton, ...	24.4905 ounces.
	Showing an extraction of silver in roasted ore by washing with water of, .	14.5616 per cent.
	Chlorination, ...	64.3261 "
	Extraction, ...	88.5743 "
First ordinary sample,	Silver per ton, ...	11.8174 ounces.
	Showing an extraction of silver in roasted ore by leaching with hyposulphite of soda of, . .	58.7738 per cent.
Final tailings-sample,	Silver per ton, ...	3.7984 ounces.
	Showing an extraction of silver in roasted ore by all treatment of, . .	86.7485 per cent.

Calculation of Silver in Tailings.

Weight of roasted ore, ...	31,775.338	tons.
Soluble salts, ...	14.083	per cent.
Weight of tailings, ...	27,300.410	tons.
Average assay of tailings, ...	3.7984	ounces.
Contents of tailings, ...	103,702.36	ounces of silver.

From the above, the following calculations are made:

Decrease in laboratory-results by washing:	
Chlorination, ...	14.6021 per cent.
Extraction, ...	1.2114 "

Results in Laboratory Compared with Results in Mill.

		Per cent.		Per cent.
Laboratory results on charge samples: .	Chlorination,	78.9282	Extraction,	89.7857
Laboratory results on washed ore samples,	"	64.3261	"	88.5743
Mill results: Extracted by hyposulphite of soda, ...		58.7738	By extra,	86.7485

Mill-Results behind Laboratory-Results.

		Per cent.		Per cent.
Based on charge-samples,	Chlorination,	20.1544	Extraction,	3.0372
Based on washed-ore samples, . . .	"	5.5523	"	1.8258

I will not undertake here to discuss the causes for the decrease in chlorination, or the changing of the silver chloride in roasted ore, to some other form which is not soluble in a solution of hyposulphite of soda, by the washing with water. A vast amount of experimenting was done during the year, and the conclusions arrived at may form the subject of another paper.

Product.

Silver was precipitated from solutions with sodium sulphide: part of the time with a polysulphide, Na_2S_2, made from caustic soda and sulphur; and part of the time with a monosulphide, Na_2S, imported from Germany in the form of crystals. The total of sulphides produced was 442,576 pounds, divided as follows:

	Pounds.		Ounces per ton.		Ounces of Silver.
Solution-sulphides,	313,417	Assaying,	3790	Containing,	593,069.12
Wash-water sulphides,	95,442	"	2875	"	136,535.66
Lead carbonates,	33,717	"	432	"	7,698.84
Total contents of product,					737,303.62

Difference between "Apparent" and "Actual" Extraction.

In noting the difference between the "apparent extraction" (*i.e.*, the extraction determined by calculating contents of tailings as against contents of roasted ore), and the "actual extraction" (*i.e.*, that determined by calculating contents of product as against contents of roasted ore), attention is called to the fact that the contents of sulphides are based on "corrected assays," that is to say, the silver-contents of scorification-slag and cupel are added to the original assay. This addition was found, on this grade of sulphides, to amount to about two per cent. The ore-contents are figured on the commercial assay by scorification, in which no account is made of silver contained in slag and cupel, the amount of which was found to be, on this ore, from 4 to 5 per cent.

Apparent Extraction.

	Ounces of silver.
Contents of roasted ore,	782,586.09
Contents of tailings,	103,702.36
Calculated contents of sulphides,	678,883.73

On this calculation the apparent extraction of 86.74 per cent. of the silver in roasted ore is based.

	Ounces of silver.
Actual contents of sulphides by corrected assay, . .	737,303.62
On this calculation the actual extraction of 94.21 per cent. of the silver in roasted ore is based.	
Differences between calculated and actual contents of sulphides = 6.78 per cent. or, ...	58,419.89

Actual Extraction.

Contents of raw ore, ...	861,488.05
Contents of roasted ore, ...	782,586.09
Loss in roasting and dust-loss (9.157 per cent.), . .	78,901.96
Contents of product, ...	737,303.62

Or 94.21 + per cent. of the silver in roasted ore and 85.58 ÷ per cent. of the silver in raw ore.

Consumption and Cost of Salt, Coal and Chemicals per ton of Ore Treated.

	Used per ton of ore. Pounds.	Cost per ton of ore.
Salt, ...	244.48	$0.8719
Coal for drying ore and salt, . . .	85.77	.1322
Coal for roasting ore, ...	117.44	.1819
Blue-stone, ...	13.45	.6331
Hyposulphite of soda, ...	9.98	.3742
Chemicals for precipitating silver, including caustic soda, sulphur and sodium sulphide, ...	18.86	.6835

The Tin-Deposits of Durango, Mexico.

BY WALTER RENTON INGALLS, NEW YORK CITY.

(Florida Meeting, March, 1895.)

VAGUE references to tin-deposits in Mexico are scattered throughout technical literature, and that country has been looked to as a likely source of a part of the world's supply of tin at no very distant date; but thus far nothing but occasional small lots have come forward, so that the notion has arisen that these mines are either mythical or unworkable. So far as I am aware, there has not yet been published any description of the tin-deposits of Mexico except a few brief references; and this paper, based on a study of them made in 1892, is therefore offered.

It is uncertain when tin was first discovered in Mexico, but it is likely that the metal was known and worked by the Aztecs and kindred tribes at the time of the arrival of Spaniards. Prescott, indeed, in his *Conquest of Mexico*, speaks incidentally of this metal as being in use among them, saying that they obtained it from the mines at Tasco. In any case there is no doubt that the mines were exploited by the Spaniards at an early date. The first exact reference to the existence of tin-ore in Mexico, however, is probably to be found in Humboldt's *Essai Politique sur la Royaume de la Nouvelle Espagne.** The author says that at the time of his visit to Mexico (1803) tin was obtained from washings at Gigante, San Felipe, Robledal and San Miguel el Grande, in the Province of Guanajuato, and between the towns of Xeres and Villa Nueva in Zacatecas. With reference to these deposits, he wrote:

"The most common mineral is the concretionary oxide of tin, or the 'wood-tin' of the English mineralogists. It appears that this mineral originated in veins traversing trap-porphyry; but the natives, instead of attacking these veins, prefer to win the mineral from the alluvial deposits which fill the ravines. The intendance of Guadalajara produced in 1802 about 9200 *arrobas* of copper and 400 of tin. . . . .†

The exports of tin from Mexico to other parts of America in 1803 amounted to 58.5 quintals, equivalent to 5,934¼ pounds.

Ward ‡ makes no mention of tin-mining in connection with his travels in Mexico; and as he was a very keen observer, it is fair to presume that there was none in his time. Andres del Rio, the eminent Mexican mineralogist, writing in the early part of the century,§ mentions the existence of tin in his country, especially "wood-tin," which, he says, was found only in alluvial deposits, but would probably be found in the mountains in veins or beds (*mantos*). Jameson asserted that it was found in Guanajuato in trap or trachytic porphyry (?), and according to Sonneschmid, the mineral (wood-tin) had so many colors that it was "easiest to enumerate those that were wanting." An analysis of mineral from Guanajuato by Collet Descotils gave 95 per cent. SnO_2 and 5 per cent. Fe_2O_3. The

* Paris, 1827, *tome* iii., 308 and *tome* iv., 76.

† 105,857.5 kilos (233,321.2 pounds) and 4602.5 kilos (10,144 pounds) respectively.

‡ *Mexico in* 1827.

§ *Elementos de Orictognosia*, 2d edition (Philadelphia, 1832), p. 211.

Fig. I.

MAP OF PART OF MEXICO. SHOWING LOCALITIES OF TIN ORES.

occurrence of tin in Mexico was also mentioned by Karsten in his edition of *Mineralogische Tabellen*, published in 1800.

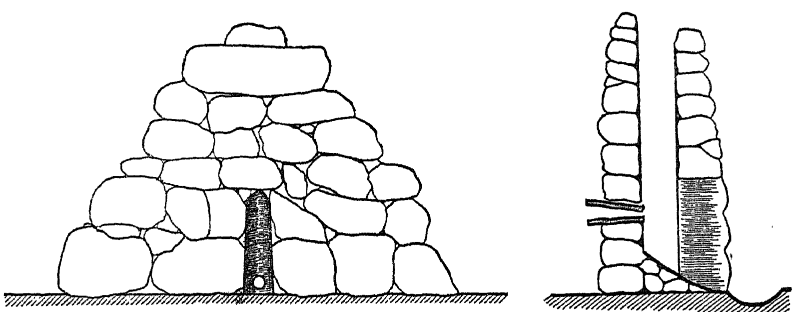
Besides the localities mentioned by Humboldt, tin has been found in Mexico at the following points: Durango, Cacária, and Potrillos, in the State of Durango; Canitas, Sain Alto, and Chalchihuites, in Zacatecas; Bolaños, Comanja, Teocaltiche, and Paso de Sotos, in Jalisco, and on the Ranchos de la Peña and San José de Estaños, in the same State; at the Cerro de Zamorano, and in the Distritos de Chichimequillas and Bravo, in Querétaro; at the Cerro del Chiquihuite, in Aguas Calientes; in the Distritos de Ures, Arizpe, and Saguaripa, of Sonora; at Catorce, Valle San Francisco, and on the Mesas de la Cruz and San José Buena-vista, in San Luis Potosi; also near the Santuario del Desierto, in the cordillera extending southeast from the capital of the State, San Luis Potosi; at Guanajuato, Dolores, Villalpando, Hidalgo San Diego, and between Santa Rosa and la Fragua, in the State of Guanajuato. This list is probably incomplete, and being made up from the casual references of travellers and others, it is likely that many of the places mentioned are imperfectly described by name and cannot be found on the map. The list is given here merely for the purpose of showing the widespread distribution of tin in Mexico. There are records of mining at only a few of these places, the most important in this respect being Cacária and Potrillos in Durango, and Teocaltiche in Jalisco.

Potrillos, which is the name of one of the outlying ranches of the great ranch of Guatimape, is situated in the Sierra de San Francisco, a spur of the Sierra Madre, about 100 miles north of the city of Durango, and 25 miles from Coneto, where some very ancient silver-mines are still worked in a desultory manner. About 12 miles northwest of Potrillos is the famous Promontorio mine, which in recent years has been one of the largest producers of silver in Mexico. The region is rough and rugged, seamed with deep and narrow ravines, or "arroyos," down which the water rushes in torrents during the rainy season. Trails have been opened through it in various directions, but, with the exception of the main road to Coneto, there are none over which a wagon can be driven. All transportation (except over this road) is done necessarily, therefore, by means of horses, mules, and burros, used as pack-animals.

The general altitude of the valleys of Potrillos is probably about 6500 to 7000 feet, while the crests of the mountain-ranges are from 1500 to 2500 feet higher. The water falling in this district drains to the north, eventually reaching the river Nazas. From the middle of July to the middle of October the precipitation is extraordinarily heavy; during the remainder of the year there is no rain, and by April the drought becomes burdensome. The region is sparsely timbered with pine, scrub-oak, and madroña, or mountain-mahogany. The two last are valueless for everything but fire-wood, but very fair lumber can be sawed from the pine.

It is uncertain when tin-stone was first discovered in this dis-

Fig. 2.



MEXICAN TIN SMELTING FURNACE
Scale, 1 in. = 4 ft.

trict, but in 1839 at least there was a considerable population engaged in getting it from the stream-beds, among them being a German colony established at Gazapera, or Sapioris, which was then the chief center of the industry. The mineral was obtained mostly from the placers, but the outcrops of certain of the veins were also worked somewhat; the Gazapera Grande mine and those on the Arroyo de Colorado (Arroyo de Estaño), subsequently known as the Grant, Candelaria, and America mines, were opened at this time. The tin-bearing gravel and the ore from the veins, which could be crushed with great ease, were washed in *bateas* and on *planillas*, the latter being simple inclined planes, set up at the edge of the stream, whence water was thrown on the ore, washing down the gangue and leaving

the heavy tin-stone behind. The method of mining the gravel was to sink pits to bed-rock in the arroyo-bottom, which was in no place very deep, and from them to dig out under the non-productive overburden. The old pits at Gazapera are now filled with gravel that has been washed in by the rains; and in many places large trees are growing in them. Here and there are the ruins of stone huts and small smelting-furnaces; but otherwise there is no evidence of former mining operations in this range, which is now given over to cattle alone.

The black tin produced in the Potrillos region was in part smelted there, and in part sent for reduction to Coneto. Its value was held to be from $1.50 to $2.50 per *arroba* (25.36 lbs.). The smelting-furnaces were small shaft-furnaces, built of uncut stone, and lined with clay. One of these furnaces, 5 feet from top to hearth, was 6 by 9 inches at the top, and 9 by 9 inches at the tuyere, which was 3½ feet below the top. Another, 4 feet high, was 9 by 7½ inches at the top, and 6 by 7 inches at the tuyere, which was 3 feet from the top. The hearth sloped sharply to the tap-hole, and metal and slag were drawn into a basin outside the furnace. This basin was nothing but a hole in the ground tamped solidly with clay. The blast was furnished by a bellows made of ox-hide, with a piece of iron pipe for a tuyere. The fuel was, of course, charcoal.

The tin-mining industry did not continue long in Potrillos. In 1846 the Indians came down from the north and drove the miners away. The region was infested by these savages until 1867, when they, in turn, were driven out by the Federal troops. During this time (in 1861), it had been visited by Captain Box and his colony of 160 Americans, in search of the lost gold-mine known by his name; but they did no tin-mining, and, indeed, after a very short sojourn, came miserably to grief. Immediately after the Indians were expelled, miners from Coneto straggled into the diggings again, and three Americans, whose names are not recorded, established a camp near Potrillos, and engaged in mining and smelting the ore. The stream-beds had been nearly exhausted, however, and there were at no time so many miners there as there had been thirty years previously. The region was still very unsettled, being overrun by roving bands of Indians, and terrorized by the famous bandit, Heraclea Bernál, and his company, who levied tribute upon the towns

and *haciendas*, and burned and pillaged until 1887, when he was killed, and his followers dispersed.* This condition of affairs explains the lack of reliable information as to the tin-deposits of this part of Mexico, at least.

The geology of the Potrillos district is simple. The prevailing country-rock is rhyolite and rhyolite-tuff, chiefly the latter, of recent origin, like that which covers vast areas in the northern and northern-central parts of Mexico.† Beyond the limits of the sheet of young igneous rocks, a gray, finely crystalline granite is exposed, and it is probable that this is the underlying

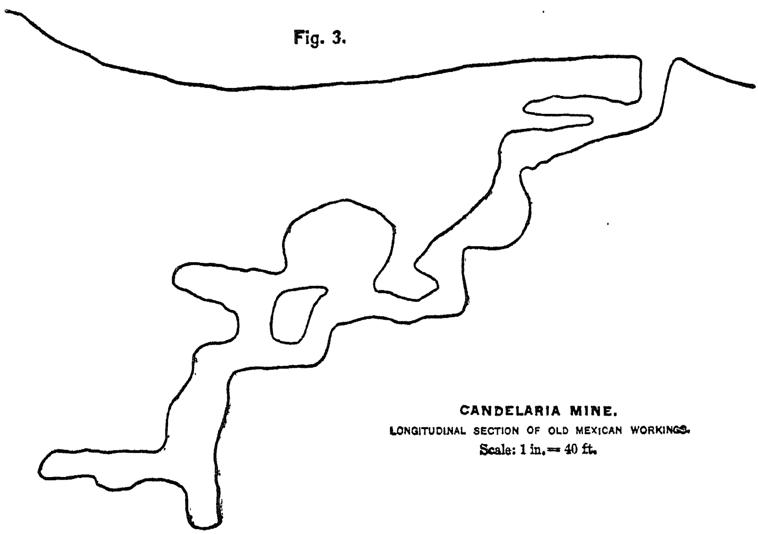


Fig. 3.

CANDELARIA MINE.
LONGITUDINAL SECTION OF OLD MEXICAN WORKINGS.
Scale: 1 in. = 40 ft.

rock. In the direction of Promontorio the Sierra de San Francisco is composed wholly of this granite, which is there intersected by numerous silver-bearing veins. Of these, however, the Promontorio is the only one that has yet proved to be rich. The rhyolite-tuff of Potrillos is generally a grayish white, soft rock, which, even at the depth of 150 feet, is very much decomposed, and upon exposure to the air rapidly crumbles. At one of the shafts that was sunk in 1891, a hut was built by certain

* The above history was obtained from inhabitants of Coneto, and miners who worked at Potrillos in 1840, and, though possibly inexact, is probably correct in substance.

† I have here to acknowledge my indebtedness to Prof. J. F. Kemp for the identification of rock-specimens submitted to him.

of the natives, who used for its walls the fresh blocks of stone hoisted from the openings underground. Soon after the summer rains began, these walls became so soft that a knife could easily be thrust into them to the depth of its haft; and finally, after six weeks or two months, they fell in a common heap of sand and clay. The rock was generally of this kind, but in the Candelaria mine changed suddenly to a hard, brittle, purplish variety, which has been pronounced a devitrified obsidian. Its color faded somewhat, however, under exposure on the surface. This rock was so hard and brittle, that the same miners who had been making easily a headway of four feet per day, were able to accomplish only one foot with difficulty. It occurs in the midst of the rhyolite-tuff in irregular masses, the one gradually changing into the other.

The ordinary rhyolite-tuff of the region shows a most striking fluidal texture, which is so marked in certain places that the exposed outcrops have the appearance of stratification. Elsewhere it is often spherulitic, with concretions as round and as large as a cricket-ball. As accessory minerals, the rock contains biotite, hematite, quartz, hyalite and chalcedony, fluorite and topaz, the last in grand crystals of various colors, and as much as 0.6 inch in length. Excellent specimens of free crystals can be found in the beds of dry water-courses on the hillsides, the soft matrix having been disintegrated and washed away.*

In many places the surface is covered with large concretions of chalcedony, usually milk-white or ferruginous, which are found in the mine openings in close association with the tin ores. The rhyolite-tuff of Potrillos also contains hyalite, but affords no such fine specimens as have been sent from other parts of Mexico to the mineralogical cabinets of New York and Europe. Another noteworthy surface characteristic of the region is the existence in certain parts of small rounded fragments of obsidian (about the size of ordinary marbles), weathered gray on

* The occurrence of topaz in the trachyte and rhyolite of Mexico has already been noted by Dana (*System of Mineralogy*, 6th ed., p. 495), who says that it has been observed in association with cassiterite, magnetite, and durangite near the city of Durango, and at La Paz, in Guanajuato, and in connection with rutile in San Luis Potosi. It is interesting to note that topaz occurs also with the tin-ores of Altenberg and Zinnwald (Saxony and Bohemia), and at Mt. Bischoff, Tasmania, and in New South Wales, though the ores of these places are widely different.

the surface, but black and shining when cracked to expose fresh faces.*

The sheet of rhyolite-tuff at Potrillos has a well-defined joint-structure, the principal planes striking in a general east and west direction and dipping south, and is intersected by a series of faults with the same strike and dip as the joint-planes. It is not unlikely that the jointing and faulting were produced at the same time by the same force. There is no way of measuring the dislocation occasioned by the fault-series, but it was probably small, the fractures being smooth and regular with no evidence of brecciation of the country-rock or in-dragging of fragments. It might be doubted, indeed, whether these apparent fractures were faults at all and not merely joint-planes, since ordinarily they are marked only by a thin selvage of kaolin not much thicker than a sheet of wrapping-paper, and the joint-planes, when freshly exposed, show a similar greasy surface; but in some places the fault-seam is as wide as 0.2 inch, with white or black clay, and in the Candelaria mine, where the rock was hard, there was actual evidence of motion in the slickensides which had been preserved. In the case of certain of the best-defined faults there was a well-marked sheeting of the country-rock on each side of the main fracture, which divided it into bands about 1 foot or 2 feet wide.

Cassiterite occurs in the rhyolite-tuff along these fault-planes in aggregations of nuggets (called *guijilos* by the native miners) and occasionally in bands of crystalline mineral replacing the country-rock between two sheeting-planes. With the former class of ore, which is the more usual, there are all gradations from nests of pure mineral down to small particles disseminated in gangue, so that the whole may not assay as much as 1 per cent. of metal. The ore that has been mined, however, has generally shown from 3 to 10 per cent. of metal, and, as thus obtained, has been broken from breasts 3 or 4 feet wide. The ore is found in pockets (*bolsas*), which are never large, occurring in a series, or chain, connected by thin stringers of ore and having a general dip to the west at an angle of about 45

* Similar specimens of obsidian from Australia were examined by the late Prof. A. W. Stelzner, of Freiberg, who held the opinion that they had been ejected from volcanic vents and imbedded in soft surface-material, as bullets falling from a gun might be.

degrees. Thus the form of the ore-bodies is represented somewhat by that of a long bean-pod. In none of the crevices, so far as opened in 1892, had more than one channel of ore been disclosed; but as the linear development had not been extensive, this was not proof positive that parallel channels did not exist. However, sufficient work had been done to show that if there were such channels, their distance from the known ones was out of proportion to the width of the latter.

These ore-bodies are exceedingly ill-defined, passing into the barren country-rock by insensible gradations. Beyond the ore-channel, or string of pockets, on the line of the fissure, there is no vein-matter, but merely a thin seam cutting the country-rock. Although narrow, this is usually well-defined upon fresh exposure, but soon becomes obliterated under the action of the air. In the Candelaria mine, however, where the country-rock suddenly changed to the hard, brittle, devitrified obsidian, as previously mentioned, there was a crevice about 12 inches wide, which was filled with white, granular quartz, although, in the soft rock immediately east, the fissure was marked only by the ordinary seam. Incidentally may be mentioned a well-defined vein of crystalline calcite, about 3 feet wide, which occurred in the neighborhood of some of the principal tin-veins, and had the same general strike and dip as they had.

The cassiterite of Potrillos has fully as remarkable characteristics mineralogically as have the deposits in which it occurs. It is found generally in mammillary or botryoidal concretions, less frequently in a crystalline form, and in the water-worn pebbles from the stream-beds takes the form of wood-tin with extraordinary striations. In color it is oftenest black, but very frequently brilliant red, and occasionally yellow. The red, crystalline variety often sparkles with such brilliancy that the name "ruby-tin" seems appropriate. In the veins the tin-ore is invariably associated with micaceous hematite, which is also found very generally in the country-rock as an accessory mineral. Dana speaks* of durangite, the fluo-arsenate of sodium and aluminum, as being found in a vein 4 to 6 inches in thickness at "the Barranca mine, 18 miles northeast of Coneto."

* *System of Mineralogy*, 6th edition (1892), p. 781; also *Ann. de Chimie et de Physique*, 5 *ième Série*, 4, 401, 1875.

There was an old mine known by this name in the Sapioris district, but it had been abandoned as valueless, and in my examination of it I failed to discover the mineral referred to; nor did I find it in any of the mines of this region. The tin-ores were, indeed, somewhat uninteresting with respect to accessory minerals. There was no evidence of wolfram-compounds either in the ore or in the metal smelted from it. A very remarkable fact was developed, however, in the latter. Two lots of 56 pigs and 310 pigs respectively, together amounting to about 25 tons, were shipped to the United States in December, 1891. Analyses of these, by Dr. Gideon E. Moore, of New York, gave the following results:

	Per cent.	Per cent.
Tin,	92.6104	90.6047
Antimony,	7.3139	9.2850
Arsenic,	trace	trace
Lead,	0.0331	0.0555
Bismuth,	0.0255	0.0276
Iron,	0.0171	0.0270
Total,	100.0000	99.9998

In no portion of the ore were independent antimony-bearing minerals distinguishable by the eye, and it was probably chemically combined with the cassiterite, as the late Dr. Genth has shown to be the case with arsenic in certain Mexican tin-ores.*

Float tin-stone was found abundantly in the vicinity of the veins, and in the arroyo-bottoms pockets of rounded pebbles were occasionally discovered. There was indeed every evidence of the existence of mineral in large quantity, and the comparative poverty of the veins when actually opened seemed somewhat incomprehensible. The explanation appeared to be, however, in the probability that the number of mineral deposits was much greater than was exposed. From the previous description of their occurrence, it is evident that exploration-work would be extremely hazardous except where there was mineral to be followed from the beginning. The surface of the country-rock was intersected by numerous parallel fractures

* Since this paper was put in type I have made an analysis of a specimen of tin from a native smelting-furnace at Canitas, which showed 6.83 per cent. antimony, the other impurities being lead and iron. The ore at Canitas resembles that of the State of Durango, and like that occurs in decomposed rhyolite and rhyolite-tuff.

which might be small fault-fissures, or might be only joint-planes. Mineral might be found in any of them, but it is obvious that the chance of such a discovery was exceedingly small, and that the uncovering of an ore-body would be more the result of blind luck than of intelligent prospecting. As a matter of fact the region had been looked over thoroughly during a period of 50 years and numerous discoveries of tin-stone had been made, but none except where there was an actual outcropping. There is no doubt that many of these small veins are not in fault-fissures at all, but in joint-planes merely. As further evidence of such occurrence of the ore may be cited the result of certain surface prospecting-work near the Grant mine. It being desired to lay bare the fault-seam that was followed in that mine in order to prospect for parallel ore-chutes from the surface, its course was located to the west by survey, and the fissure was then exposed by costeaning. Rounded fragments of tin-stone were found in the surface-loam (about one foot deep), and especially in the thin layer of hard clay which rested on the bed-rock and was simply the decomposed surface of the latter; but this float-mineral extended above the fault-crevice, and was followed by a trench several hundred feet long over the top of the hill without finding the source whence it came. The mineral found in this trench varied from the size of a small pea to pieces weighing 2 pounds. All were more or less smooth and rounded, but this was not necessarily evidence of attrition, the botryoidal and mammillary concretions found in the mines showing the same characteristics. The only inference, therefore, was that these fragments had resulted from numerous veinlets in the country-rock, which had been worn away. It is this which has occasioned the wide distribution of float and stream-tin out of proportion to the deposits in place that have been uncovered.

The physical conditions of the region, however, are opposed to the accumulation of large alluvial deposits that might be profitably worked. The hill- and mountain-sides are, in most parts, exceedingly steep, while the ravines are deep and narrow, the water rushing through them during the rainy season in torrents that move huge boulders. Under these circumstances there was no opportunity for the formation of broad

and thick deposits of gravel. There is undoubtedly much mineral in holes between boulders in the stream-bottoms, but it would be very difficult to recover this on an industrial scale. As for the drift-deposits on the hillsides, they are mostly too poor to pay for the expense of bringing water to them. The alluvial diggings of the region, which were formerly productive, were chiefly in the part known as Sapioris, or Gazapera; and these are now exhausted. In short, the best evidence of the present poverty of these placers is the fact that they were abandoned by the native miners, who were born and brought up in the vicinity, were perfectly familiar with the ores and understood the method of smelting them in furnaces which could be constructed at an expenditure of only their labor; and yet, during the hard times that have prevailed in this part of Mexico recently, they have made no attempt to re-engage in this industry.

There is some evidence (but hardly enough to warrant the conclusion) that the source of the tin-stone of Potrillos existed as an original constituent in the country-rock. Repeated tests of the country-rock selected at considerable distances (up to 100 feet) from any known ore-body, showed small amounts of tin, occasionally as much as 0.30 per cent. of SnO_2, but usually less than 0.01 per cent. These tests were made by crushing the sample very fine, and concentrating mechanically 500 or 1000 grammes. A curious and noteworthy occurrence of mineral was discovered in driving a cross-cut to intersect the Grant vein. At a distance of about 100 feet from the latter, no ore having been observed in the vicinity except as frequent small traces of tin in the country-rock, a bunch of ore weighing about 30 pounds was encountered. A sample of it assayed 15 per cent. of tin; the country-rock immediately surrounding it gave about 0.2 per cent. There was no evidence of a fissure of any kind at this place, and the bunch of ore mentioned was all that was discovered there.

Bischof has proved that tin-stone can be dissolved in water containing alkaline carbonates, whilst Kjerulf and Daubrée have demonstrated that it may be deposited either from solutions or from gaseous emanations.* Whether, in the case of

* A. G. Charleton, *Tin, Methods of Mining, Dressing and Smelting*, etc., 1884, p. 5.

the Potrillos deposits, the original source of the mineral was in the country-rock or not, it was apparently deposited in parts of the country-rock near small crevices which allowed the solutions to pass. These crevices were in some cases fault-fissures, and in others merely joint-planes.* Such occurrences as the small bunch of ore found in the Grant cross-cut can, of course, be regarded only as local segregations.†

After Potrillos, the next extensive developments in Mexican tin-fields have been carried out in the Cacária district, about 40 miles north of the city of Durango. These mines were first discovered about 1870 (possibly two or three years earlier) by native prospectors, who showed their specimens to certain Potrillos miners, and thus induced the latter to visit the locality. In 1873 the tin-bearing area was secured by Mr. Marcus Ison, of Durango, under what is known as the Rio Verde concession, covering 58 square leagues.‡ In 1881 this property was taken in hand by the Durango Tin Mining Company of St. Louis, which explored it on an extensive scale during that year and the next, when the enterprise was abandoned as unprofitable, and the mines reverted to their original owner. Since that time they have been worked only in a desultory manner by natives chiefly, who have smelted in their rude furnaces a small amount of metal for local consumption.

The Cacária mines are situated in the southern extension§ of the Sierra de la Candela, a spur from the Sierra Madre. The physical characteristics of the district are identical with those of Potrillos, and the geological features also are practically the same. I did not visit this district, but have received information concerning it from Mr. H. Winninghoff, who was in charge of the developments made by the Durango Tin Mining Com-

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* Mr. G. F. Becker, in the discussion of his paper on "The Torsional Theory of Joints," Virginia Beach Meeting, February, 1894 (*Trans.* xxiv., 865), refers to ore on joints in certain quicksilver-mines. The joints in the rhyolite-tuff of Potrillos are similar actual partings of the rock.

† With respect to the origin of these tin-ores reference should be made to a recent paper, "Ueber die durch pneumatolytische Processe an Granit gebundenen Mineral-Neubildungen," by J. H. L. Vogt, in the *Zeitschrift für praktische Geologie* of December, 1894, in which the tinstone in granite and acid eruptives is attributed to gaseous emanations.

‡ The Mexican *legua*, or league, is equivalent to 4.19 kilometers, or 2.604 miles; the square league contains 4339.4 acres.

§ Called variously Sierra de Cacária and Sierra de Canatlan.

pany in 1881–82. The substance of his communication is as follows:

"The formation of the entire mountain-range in which the tin is found is quartz-porphyry. The veins are irregular fissures filled with red clay (decomposed country-rock), disseminated through which a generally very impure tin-ore is found in grains, scales, and nuggets. Associated with the cassiterite are found arseniate of lead and molybdenum and tungsten minerals. The vein-matter does not contain on an average over 15 per cent. of ore, which may not give more than 25 per cent. SnO_2.*

"Stream-tin was found in a number of *arroyos* of small extent, but the richest and most accessible of the alluvial deposits were long since exhausted by native miners. Those which remain cannot be easily worked on account of the scarcity of water, there being only a few streams, and those insignificant. On account of the natural conditions it would be hardly practicable to dam up the water during the rainy season."

The most extensive developments in this district were made in the Diablo mine, which was opened to a depth of 273 feet.

The results of an examination of the ores of the Cacária district, made in 1885 with characteristic thoroughness by the late Dr. F. A. Genth, were published in the *Proceedings of the American Philosophical Society*, xxiv., 1887, p. 23. Two paragraphs from this paper, containing a description of the mines as communicated by Mr. John L. Kleinschmidt, may well be quoted here.

"The central body of the Sierra de Canatlan, in which the mines which have furnished these ores are situated, consists of quartz-porphyry, which in some places is traversed in a net-like manner by small veins of tin ores. About one mile from the Mina del Diablo doleritic rocks occur.

"The vein of the Mina del Diablo has been traced for about one mile in length, has a thickness of from 18 inches to 2 feet, is almost perpendicular, and perfectly separated from the porphyry by argillaceous selvages. It consists of a decomposed, white clayey material containing druses of quartz with tin ores"

Dr. Genth reports analyses of Mexican tin-ores as given on page 161.

Dana is of the opinion that the oxides of zinc and arsenic in these ores are to be considered as merely impurities, and not as constituting distinct mineral species.

The third important scene of tin-mining in Durango, is the Cerro de Iglesia de los Remedios, less than a half mile west of

* This is equivalent to a tenor of only 3.75 per cent. of black tin in the vein-matter. Compare with the yield of Potrillos ore. The delusion as to the extraordinary richness of the Durango tin-fields is largely due to the assay of specimens of the remarkable nodules of cassiterite found there.—W. R. I.

	I.	II.	III.	IV.	V*a*.	V*b*.	VI.	VII*a*.	VII*b*.	VIII.	IX.	X.
Specific gravity	6.820	6.594	6.911	6.535	6.714	6.712	6.581	6.160	6.219	6.509	6.199	6.496
SnO_2	92.84	93.98	93.01	92.09	86.99	86.81	92.26	84.20	84.30	92.50	89.90	93.13
Fe_2O_3	4.12	5.62	5.82	5.45	11.56	12.73	4.58	1.31	1.55	0.22	0.10	0.20
As_2O_5	trace	....	....	2.11	trace	....	1.25	9.85	10.34	4.56	5.80	3.18
CuO	trace	....	0.07	....	0.11	....	trace	trace	trace	0.16	0.20	0.09
ZnO	....	....	....	....	....	....	0.57	3.05	2.96	1.89	2.43	2.71
SiO_2	2.70	0.23	1.07	0.66	0.57	0.52	0.44	0.35	0.30	0.24	0.55	0.43
Ignition	0.34	0.24	0.27	0.07	0.20	0.34	0.26	0.39	0.57	0.26	0.40	0.32
Total	100.00	100.07	100.24	100.38	99.43	100.40	99.36	99.15	100.02	99.83	99.38	100.06

I. Bright red; from Durango.
II. Brick red; from Coneto (Potrillos).
III. A large pebble, resembling compact hematite; from Durango.
IV. Dark brown; from Coneto (Potrillos).
V*a*, *b*. Bright brick-red; from Mina del Diablo, Cacária.
VI. Dark brown; from Guanajuato.
VII*a*, *b*. Yellow; from Mina Varosa (probably this is an error for "Barrosa"), Cacária.
VIII. Yellow; from Mina Varosa, Cacária.
IX. Brownish yellow; from placers, Cacária.
X. Brownish yellow; source not stated.

the center of the city of Durango, and within sight of the famous Iron Mountain. At this point a shaft was sunk some 60 feet to open a vein of cassiterite, and longitudinal openings were made in the customary Mexican fashion; but the vein proved to be small, and the mine was soon abandoned. The country-rock is rhyolite tuff, and the vein-formation is identical with that of Potrillos; hence, an extended description is unnecessary. With respect to other deposits of tin in Mexico there is not much reliable information. Attempts have been made within the past few years to work those at Sain Alto, and certain others in Guanajuato, but with what success has not been reported so far as I know. The mines in the Canton of Teocaltiche, in the State of Jalisco, were formerly exploited somewhat extensively. According to a report to the *Sociedad Mexicana de Mineria*, by the Superior Gobierno del Estado, July 5, 1883, the mine "Los Vallecitos," in that district, had been opened to a depth of 30 meters, and for a length of 200 meters on the vein. Its product, when at its best, was about 9000 pounds of black tin per month, from which about 6000 pounds of metal were obtained. The mines at Bolaños, also in Jalisco, are still worked in a desultory manner to supply local requirements.

Before concluding this paper, mention should be made of a brief description of the Mexican tin-deposits published twenty-five years ago. It occurs in a foot-note to Baron von Richthofen's paper, "Ueber das Alter der goldführenden Gänge und der von ihnen durchsetzten Gesteine."* In speaking of the denudation of veins, the author says:

"The existence of tin in the State of Durango has been known for a long time. The Indians have been wont to gather the ore and to use the metal smelted from it for barter in the place of money. Mr. W. Ashburner made a very careful examination of the deposits which are widespread, and declared that for various reasons they are unworkable. The tin-ore is found in a hilly part of the high plateau, poor in timber and water, which consists almost entirely of trachyte. The most comes from secondary deposits of alluvial gravel in ravines and hollows. It occurs in small kernels and nuggets like the Bohemian wood-tin. It is also found, however, in original deposits in fissures (*Spalten*) in the trachyte, in part as incrustations of the walls and in part in rounded lumps of various size, which, together with small, very perfect crystals of topaz and fragments of trachyte, are imbedded in a clayey mass which fills the fissure. In the vicinity of the ore-bodies the trachyte is very much decomposed. According to hand specimens which Mr. Ashburner brought with him, it is of an ash-gray color,

* *Zeitschr. der d. geol. Gesellschaft*, Band xxi., 1869, p. 723, *et seq.*

porous and full of small spherulites; it contains black mica, and, very sparingly, small crystals of feldspar, which could not be more exactly determined on account of the advanced decomposition of the rock. Another variety has an amygdaloidal structure and contains in irregular-formed cavities a white, spongy substance, such as is very often found in trachytes and rhyolites when they are decomposed by solfataric action. I saw several pieces of the first variety, which still showed a crust of tin-stone. Mr. Ashburner had broken them from the walls of the fissure. These crusts had the peculiar kidney-shape, smooth surface of botryoidal hematite. It consists of concentric, radially fibrous layers. Any one who has seen it will recognize the unquestionable origin of the stream-tin. Since not merely one but many veins of the character described were examined, and the trachyte in the neighborhood of all of them was strongly decomposed, the recent origin of this tin-ore may be accepted as a fact, and its origin is scarcely to be sought in anything but solfataric action."

The igneous rocks of Mexico have been but imperfectly studied, and the following description of them by Prof. Don Mariano Bárcena is worth quotation: *

"Igneous granite occurs in many parts of Mexico frequently under circumstances analogous to those of the porphyries and trachytes, with which rocks it is related. This granite contains black mica, while many of the porphyries and trachytes also contain mica, together with quartz, and form insensible gradations from the igneous granite.

"The volcanic porphyries show crystals, generally light-colored, disseminated in a feldspathic paste. Sometimes they show a columnar structure like basalts. In color they are usually white, grayish or reddish, but sometimes pass into the dark colors of basalt. In hardness they vary from 5 to 6, and in density from 2 to 3.

"These volcanic porphyries of Mexico often present characteristics indicative of hydrothermal origin, or at least indicating that when they appeared they were plastic because of the water that impregnated them, since there are to be observed in them many undulating lines of hydrosilica and other substances which seem to mark the direction of fluidal currents. Thus there are to be noted in these porphyries concretions of opal, hyalite and also oxides of iron and tin, which are, without doubt, of hydrothermal origin.

"The trachytes of Mexico began to appear in the Cretaceous age and there were large outflows in the Tertiary, and the same occur even in the recent period. They are exceedingly common and are related to the porphyries previously described. The lavas of the last eruptions of the volcanoes Ceboruco and Colima are pitchstone-trachytes."

The Florida Rock-Phosphate Deposits.

BY G. M. WELLS, OCALA, FLORIDA.

(Florida Meeting, March, 1895.)

A VIEW of the map of Florida† shows the phosphate-deposits to lie on the western side of the State, extending southward over

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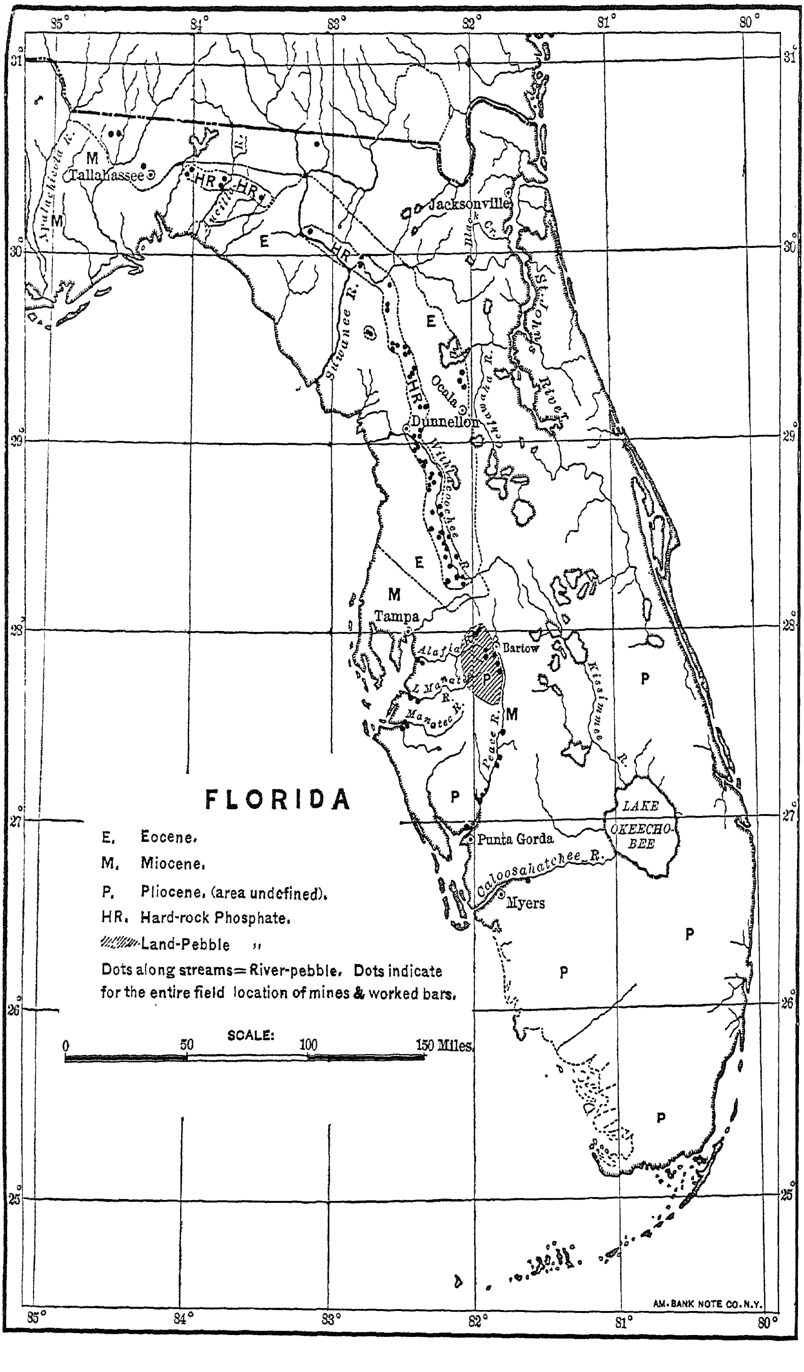
* From *Tratado de Geologia*, p. 189, *et seq.*

† See Figs. 1 and 2, of which Fig. 1 is reproduced from the paper of Mr. G.

FIG. 1.


FLORIDA
E. Eocene.
M. Miocene.
P. Pliocene. (area undefined).
HR. Hard-rock Phosphate.
Land-Pebble "
Dots along streams = River-pebble. Dots indicate for the entire field location of mines & worked bars.
SCALE:
0 50 100 150 Miles
Tallahassee
Jacksonville
Ocala
Dunnellon
Tampa
Bartow
Punta Gorda
Myers
LAKE OKEECHOBEE
St. Johns River
Suwanee R.
Withlacoochee R.
Ocklawaha R.
Peace R.
Kissimmee R.
Caloosahatchee R.
Manatee R.
Alafia
Black Cr.
AM. BANK NOTE CO. N.Y.

an area about 200 miles long by 20 miles wide. In the different portions of this area the phosphates vary somewhat in character of formation and in quality, and there are intervals containing several mines between the general groups of deposits. The mining-field of each of these general groups is made up of small deposits or pockets covering from one-eighth of an acre or less to three-quarters or one acre, with occasional pockets broken by narrow "skips," covering two to four acres. A few larger tracts of continuous deposits have been discovered; but generally these superficially larger deposits are found not to extend to as great a depth as the smaller ones; and, in most cases, the material is of lower grade and lies imbedded in an irregular formation of lime, which is an obstacle to systematic and profitable mining.

The hard-rock phosphate territory is distributed about as follows:

The Luraville tract is an apparently separated or isolated territory, a few miles northwest of the general trend of the phosphate-field, with its western margin marked by the Suwanee river. The extent of this tract is not at present determined.

Outside of present mining work, little prospecting or examination has been carried on in it; but superficial evidences, such as are considered important and favorable at other points in the phosphate-region, indicate that very large quantities of material can be produced from lands in this locality, covering an extent of three or four miles square, and that the grade will be well above the established standard of commercial requirements. The formation presents a massive exposure of mixed boulder- and gravel-phosphate wherever development and exploitation have been made. The only mining operations now conducted in this district are those of a company of French investors, which has the largest plant engaged in phosphate-mining in this State, and when its plans for operating are fully matured, will probably be the most extensive producer in the whole phosphate-territory. Eight double log-washers, with their supplement of mining appliances and drying- and screening apparatus and all that goes to make up a complete mining

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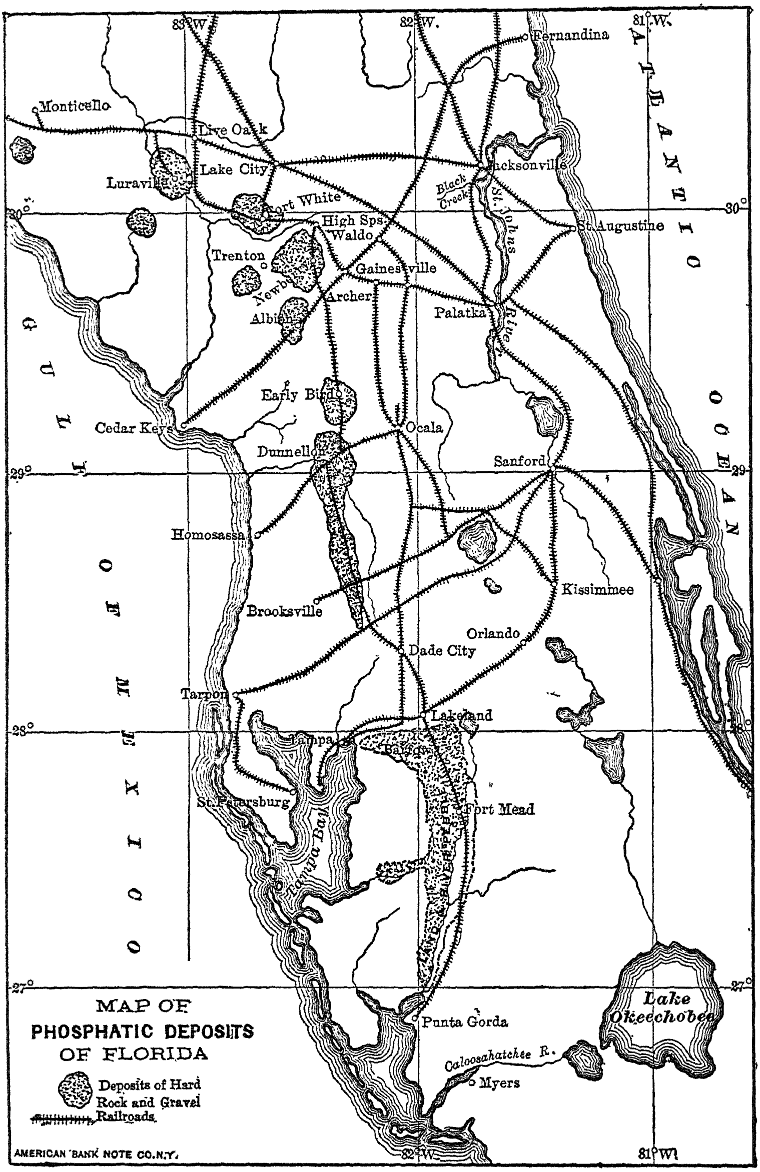
H. Eldridge, *Transactions*, xxi., 196, and Fig. 2 is here added, as showing railroad lines and some other details.

enterprise, comprise the equipment of this model and progressive French establishment.

Fig. 2.



The High Spring and Trenton section, which occupies the northern workable limit of the field of operations, embraces about

eighteen mining plants, and is estimated to produce 400 tons per day. The peculiar feature of this section is the occurrence of large pinnacles or upright boulders of limestone, which, in many cases, extend to the surface and appear as outcrops. The phosphate-deposit, which lies intimately connected with these "lime points," and is imbedded between and around them, has constantly yielded a large and profitable amount of good marketable material, easy to excavate and economically handled; and this field is considered a favorable one for operating; in fact, a number of its mines are prominently known as among the best producers in the State. In one case known to the writer, 80 to 100 tons per day is taken out and prepared for shipment at the present time, by a plant of only ordinary capacity, and with facilities that might be much improved. A new French syndicate is now being organized that will operate on a large scale in phosphate-lands near Newberry, in the High Spring section, which have been reserved from sale until the present time, and are known to be of great value. The Trenton addition with its very extensive and valuable beds of "plate-rock" phosphate will also be operated by the syndicate mentioned.

Next comes the Albion district, with a smaller area but a good product of uniform grade. About ten mining-plants are operating in this section, with an average total product of about 300 tons per day. Mining is here conducted with special difficulty, as water is reached about 25 feet from the surface of the ground. Pumping- and dredging-appliances are now being introduced with advantage, and are likely to be generally adopted in this vicinity, certainly after a depth of more than 20 or 25 feet has been reached. Dredges of the ordinary bucket-pattern are constructed at the margin of the mines, and launched into the water, when sufficient depth has been attained to float them, the processes that follow being the same as those employed in dry mines in other localities.

South of this field is what is known as the Early Bird section, covering a large area. Here the beds of phosphate are found very commonly intervening between bodies of sandstone and of boulders of flint, and lying upon a bed of limestone; the phosphate-deposits being located in the basins or indentures between these flint tracts. Here most of the phosphate material is imbedded in clay, and presents difficulty in the prepa-

ration of a clean material. This is now being overcome by a more thorough mechanical handling; and the production of high-grade phosphate in this section is about 250 tons per day; five or six plants being at work, and others in process of construction.

The next is the Dunnellon district, which, with the Piedmont, Alachua, Stonewall, Marion and Tiger additions, is the original center of the hard-rock mining-region. The first mining of hard rock in Florida was begun at this point; and the mines here have been constantly producing phosphates of the best grade since the discovery of these valuable deposits. The mines at Dunnellon are worked on a more extensive scale than any of the other mining properties in the State; their product and methods of mining have given a character and impetus to this special industry; and the name of this district has carried with it a guaranty of value, and has been an important factor in promoting the phosphate business in this State. At these mines and in the immediate vicinity there are mined and prepared for shipment about 350 tons per day of phosphates of high commercial value. The machinery, appliances and devices for handling material used in this district, are of the most modern type, and the system of labor is generally conceded to be unexcelled.

The Citrus county phosphate-field is a district, about 40 miles in length and varying in width from one to five miles, with a large number of separate deposits of phosphate, nearly all of which extend to more than the average depth of the material as generally found elsewhere. It presents largely a boulder-formation of very high grade, which can be taken out and handled at moderate expense for cleaning and preparation for market. The boulder-formation generally lies in a so-called gravel-phosphate matrix or covering, and the yield of good material from these deposits, when the matrix is manipulated by mechanical appliances, shows probably the largest percentage of the bulk of crude product mined that is obtained in the State.

The Anthony deposits, in Marion county, near Ocala, were considered at the very beginning of the phosphate-discoveries an important workable field of operations, and a number of plants of varying and peculiar construction were started in this

territory. Unfavorable conditions, however, prevailed in this section, and some operators failed in their mining enterprises. The district may be looked upon as the experimental field or school of experience; failures here have given rise to the knowledge of better methods and the introduction of improved appliances in other places. The Anthony field, under more favorable auspices, is still likely to be an important factor in the Florida phosphate-production.

The whole of the territory thus sketched and indicated on the phosphate-maps of the State is included in the high-grade hard-rock district, and has produced, since the discovery of phosphate in 1890, about 900,000 tons of marketable material, analyzing from 76 to 82 per cent. of bone-phosphate of lime. The present production in this hard-rock field is about 1400 tons per day. About seventy mining plants (some of very small capacity) have been erected in this territory. Probably fifty of these plants are at present in full operation, while the others are in process of remodeling to comply with newer methods, or are being placed in better positions for successful work.

There is much difference in the net yield of good material in the several mining fields mentioned. A fair output for a plant consisting of a double log-washer, with boiler, engine, pumps, screening-machines, dryers, etc., is 30 to 40 tons of cleaned and dried product per day. The apparent discrepancy between the output of mines as stated above in connection with the different sections, and the estimated product of about 300,-000 tons for the present year from the whole hard-rock field, is accounted for by the voluntary suspension of work, at some mines, by reason of changes of machinery, repairs, and other conditions common to all mining enterprises, and no doubt familiar to those members of the Institute who are engaged in active mining.

The cost of mining depends upon the amount of overburden or superficial earth to be removed before reaching the phosphate, and the percentage of good material contained in the bulk of the material that is carried to the washer. Some of the deposits begin as outcrops; but the greater portion of the phosphate is taken out beneath from 6 to 15 feet of overburden —partly sand and partly a clayey loam, and, in many instances,

a stiff tenacious clay; in which case the labor and expense are greatly increased. The average depth from the surface to which the deposits extend is about 50 feet, the last few feet being generally worked below the natural water-level of the phosphate-field. All mining is done in open cuts.

The present price of phosphate is but little more than one-half what it was at the beginning of the business in 1890 and 1891, but the production is steadily increasing, and the cost of production has been so much reduced by the use of modern machinery, by a better knowledge of mining methods, and by an enforced economy in expenditure, that, even at the present low prices obtainable for phosphates, the business is fairly remunerative. With a better condition attending general business throughout this country and abroad, mining operations in Florida may be expected to improve and become again a very profitable industry.

There is no phosphate region in the world, known to-day, that possesses so many advantages for successful mining as that of the Florida deposits. The grade of the material shows the highest average that is produced anywhere. The facilities for moving the material to points for distribution are good. The average distance from mines to ports of shipment is about 150 miles. The distributing ports for the hard-rock districts are, Port Tampa, Fernandina, Brunswick, and Savannah, the largest tonnage being moved from Fernandina, where storage-bins are located and loading facilities are good. Port Tampa, the terminus of the Plant System of railroads, is constantly adding facilities for the prompt handling of cargoes of phosphate, and at present nearly equals Fernandina in the amount of its shipments. Railroads are numerous and can be cheaply constructed when necessary to extend them into new sections. The machinery needed to mine and prepare the material is simple and inexpensive compared with that generally used in other mining operations; and the cost of a plant with sufficient land to work upon is within the reach of small investors. The working days at the mines are about two hundred and eighty during the year. The climate is healthful; labor is readily obtained at a fair compensation, and skilled operatives are at hand who are becoming familiar with the business. The mining camps are generally well regulated, and proprietors and employees can re-

side at the mines with safety and with little inconvenience, as supplies of all kinds can be readily obtained at the towns located in the near vicinity of all the large mining fields. Telegraph- and mail-facilities are within easy access of nearly every mining-camp in the State.

The mine-operators are mostly persons who have been drawn from other vocations, but are men of high character; are persevering and energetic, have profited by hard-earned experience, and have become generally proficient in their new employment, and successful in adapting means to ends, as is shown by the rapid and still-increasing growth of the industry.

Florida phosphates are mostly shipped to European ports, and are manufactured into fertilizers in England, Ireland, Germany, and France. Quite recently, shipments have been made to the Sandwich Islands. Foreign agents of consumers and dealers in phosphates have their offices near the centers of production, and contracts for delivery and prices are commonly made at points of shipment, the material being sold at a given price per unit of its content of phosphate of lime. The Florida phosphates are all used in the manufacture of commercial fertilizers and super-phosphate.

There has been much speculation as to the extent of the phosphate-deposits of Florida. Government officials and others have made investigations and reports on the subject, with widely varying conclusions. As new lands are being developed and worked, our knowledge becomes somewhat enlarged, and it might now be within reasonable limits to say, that the hard-rock phosphate regions of Florida embrace about 500,000 acres, and that there may be 500 acres of actual phosphate-deposits which could be estimated to yield 20,000 tons per acre, making about 10,000,000 tons of hard-rock phosphate as a possible amount that may be reckoned as available for future supply. There is a steady and constantly-increasing interest in this industry; new mining-plants are still being erected in all parts of the phosphate-region, and the capital which is invested in this business, now amounting to more than $10,000,000, will probably be very considerably augmented in the near future.

Ocala, in Marion county, a town that contained in 1868 a population of 200 or 300, has become a thriving and prosperous city, with modern improvements and conveniences, and is

the center of the hard-rock phosphate-industry, with good banking and other financial facilities. Railroad-connections and transportation-arrangements are complete and suited to the increased volume of business resulting from the phosphate-developments, and the place is well adapted in every way for the homes and offices of investors and operators employed in such enterprise.

No reference has been made in the above statement to the land- and river-pebble phosphates of the more southern part of the phosphate-belt, which are called the "South Florida pebble-deposits," and are composed of a drift-formation of small pebbles from the size of grains of wheat to that of a walnut. These deposits lie nearer the surface than the hard-rock phosphate, and are of variable depth and thickness, covering larger continuous areas than those of the hard-rock regions. The percentage of bone-phosphate of lime is less than that contained in the rock of the territory above referred to, being an average of about 60 to 65 per cent.

The mining work is mostly done by hydraulic processes, pumping-plants being so arranged as to wash down the mass of pebble and matrix, and lift the whole to elevated washers and screens for separation and cleaning, after which it is returned to revolving cylindrical dryers, and from these to storehouses, ready for shipment. Powerful dredging-machines are also used in mining operations in the South Florida deposits, both in the land- and river-mining, and the machinery used is generally much more complicated and expensive than that in the hard-rock phosphate-section.

About 600,000 tons have been mined and shipped to American and foreign ports from the "pebble" district since the beginning of mining there. Port Tampa, on Tampa Bay, and Punta Gorda, at Charlotte Harbor, are the principal shipping-points for the pebble-phosphates. There are 15 plants engaged in the production of this material, and railroad- and water-facilities are good and convenient for handling the product. The undeveloped area of phosphate-lands in this section is large, and a continuance of the amount at present produced may be expected for several years, if conditions of demand and prices are favorable.

The Ducktown Ore-Deposits and the Treatment of the Ducktown Copper-Ores.

BY CARL HENRICH, DUCKTOWN, TENN.

(Florida Meeting, March, 1895.)

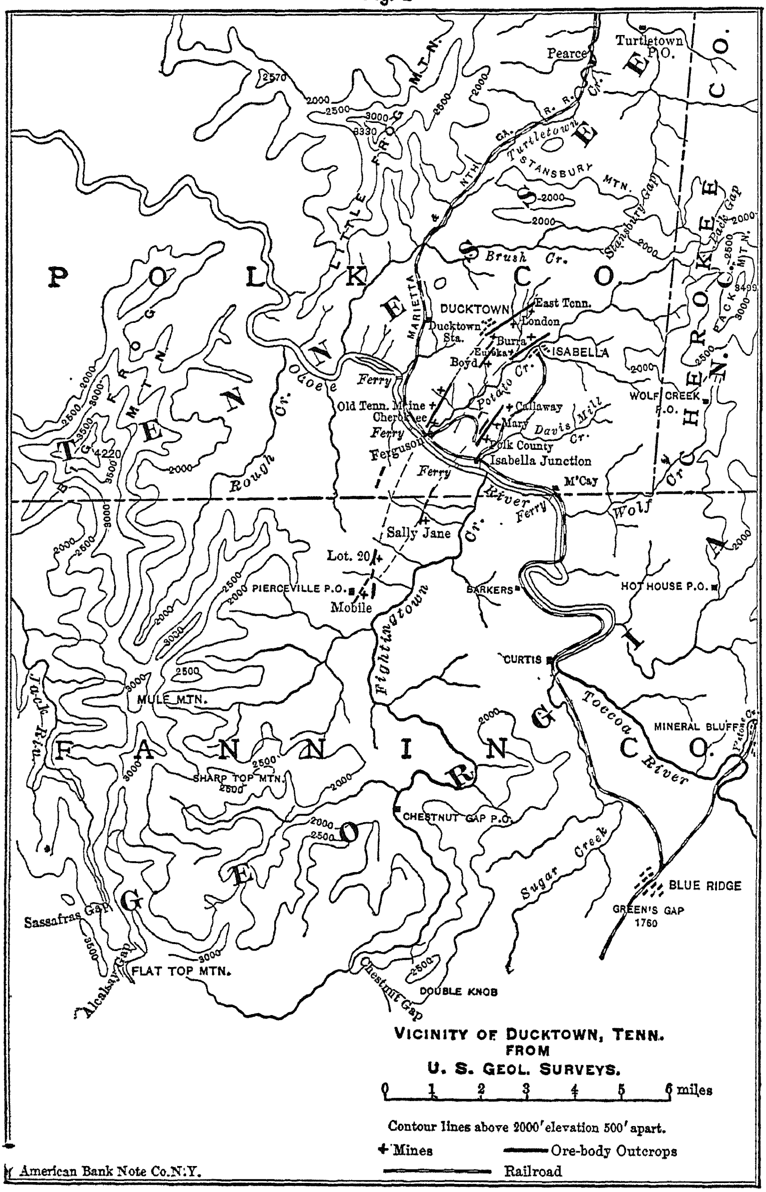
I.—Geography and Topography.

The Ducktown copper-mines are located in the southeast corner of Tennessee. The name Ducktown was originally given to a district occupying the southeast corner of Polk county, which in turn occupies the southeast corner of the State. Fig. 1 is a map of the vicinity of Ducktown, taken from the topographical surveys of the United States Geological Survey. The Ducktown district is bounded on the east by Cherokee county, North Carolina (the boundary-line between the two States running here a little east of north along the western slope of Pack mountain), and on the west by the eastern foot-hills of Little Frog mountain. On the north, Stansbury mountain, really a chain of low mountains or hills, forms the dividing-line between Ducktown and Turtletown districts. On the south, the Ducktown district extends beyond the Ocoee river to the Georgia State line. This line, in going west, gradually recedes southwards from that river into the foot-hills of Big Frog mountain, the highest mountain of the neighborhood, which rises about 4300 feet above sea-level, while the surface of the Ocoee river, where it leaves the Ducktown district, is less than 1500 feet above sea-level.

It will be seen from this description, and from a study of the map, that the Ducktown district occupies the northern part of a kind of basin, bounded on the east by Pack mountain and on the west by Little and Big Frog mountains. Between these mountains the Ocoee river, draining the basin, cuts its way to the Hiwassee and the Tennessee rivers in a narrow gorge, through the crystalline slates and schists, which from their excellent exposure in this Ocoee gorge have received the local

name of the Ocoee formation, and are dubiously assigned to the Middle Silurian by the United States Geological Survey.

Fig. I.



VICINITY OF DUCKTOWN, TENN. FROM U. S. GEOL. SURVEYS.

Pack mountain, with its lower continuation to the north and south, and also the Frog mountains, belong to the series of

parallel mountain-ranges which everywhere, from Alabama to Canada, form a prominent characteristic feature of the Appalachian mountain-system. The axis of these ranges runs about N. 20° to 25° E. in the vicinity of Ducktown. Stansbury mountain, however, the northern boundary of the basin, has its axis nearly at right angles to that direction. This range of hills had its origin in a folding and faulting of the strata, acting in a direction nearly at right angles to that main force which is considered as the cause of the uplift of the many parallel mountain chains, running in a general northeast direction and forming together the Appalachian mountains. I have been informed by members of the United States Geological Survey that indications and proofs of this secondary folding and faulting, at nearly right angles to the main uplift of the range, have been found by them at many points in the Appalachians. I shall have occasion, later on, to recur to this point.

The basin of which Ducktown forms the northern portion is cut up by many larger and smaller creeks, fed by a plentiful supply of springs. These creeks have formed erosion-valleys, narrow where they cut through the harder rocks and widening out where the rocks are softer. Between these numerous valleys lie rounded hills and ridges, the tops of which rise from 1700 to 1800 feet above the sea-level, while the valleys are from 100 to 200 feet lower as they descend to the Ocoee river.

The town of Ducktown, or Hiwassee Town, as it was formerly called by the old inhabitants, occupies one of the highest points of the basin, near the center of the mining district. As the name of the post-office is Ducktown, this name has been gradually applied to the town in place of Hiwassee. Ducktown is also the name of the station (locally known as Simstown) on the Marietta and North Georgia railroad, at which the mail for and from Ducktown post-office, nearly two miles distant, is delivered.

II.—History of Ducktown.*

Before Ducktown was inhabited by any white man, and even before the Cherokee Indians held possession of its forest-clad

* This history of the Ducktown copper-mines has been compiled by the writer

hills, this part of the country had probably been the scene of the mining and smelting industry of a race of which our only knowledge is derived from the remnants of their former villages and mounds, containing tools and domestic utensils, especially pottery, more or less well preserved. That these mound-builders were familiar with the use of copper, and that the Lake Superior native copper was mined and used by this prehistoric people, has long been believed. But that they were able to extract metallic copper from its ores, and that they mined and smelted such ores, is a fact new to the writer, and probably likewise to many members of the Institute.

About 1880, after an unusually severe freshet, Judge James Parks, of Ducktown, found, near the mouth of Davis Mill creek, where that stream empties into the Ocoee river, a quantity of Indian relics—pottery-fragments, arrow-heads, etc.—uncovered by the rain in an old field. To his surprise, however, he found also, besides these Indian relics, a few pieces of rich carbonate copper-ore and pieces of slag or cinder, besides many fragments of pots, which appeared to have been lined with a charred mass. Some of the slag-pieces were of peculiar shape, as if they had cooled adhering to the upper rim of a crucible, after the main contents of the crucible had been poured out. Judge Parks's father was one of the first settlers of the region, and is thoroughly familiar with all that has happened in the region since the advent of the white man. These ores and slag-remnants, and parts of artificially-cut square fire-proof stones of oblong shape (one of them glazed and sintered on its head from the effect of fire), were not brought to that spot by white men; nor was a small irregular slab of metallic copper, and remnants of Indian (or mound-builders') pottery, obtained afterwards in 1894 by Judge Parks, at the solicitation of the writer, from the same spot.

We have here, apparently, a perhaps unique remnant of a former metallurgical industry, that of melting copper from its oxidized ores, practiced by the mound-builders, who probably

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from the accounts furnished to him by many of the older inhabitants, among whom Judge James Parks, E. M. Kilpatrick, and William Jory ought to be mentioned, and from J. B. Killebrew's *Report on the Ocoee and Hiwassee Mineral District*, Nashville, 1876, and Arthur F. Wendt's *The Pyrites Deposits of the Alleghanies*, New York, 1886.

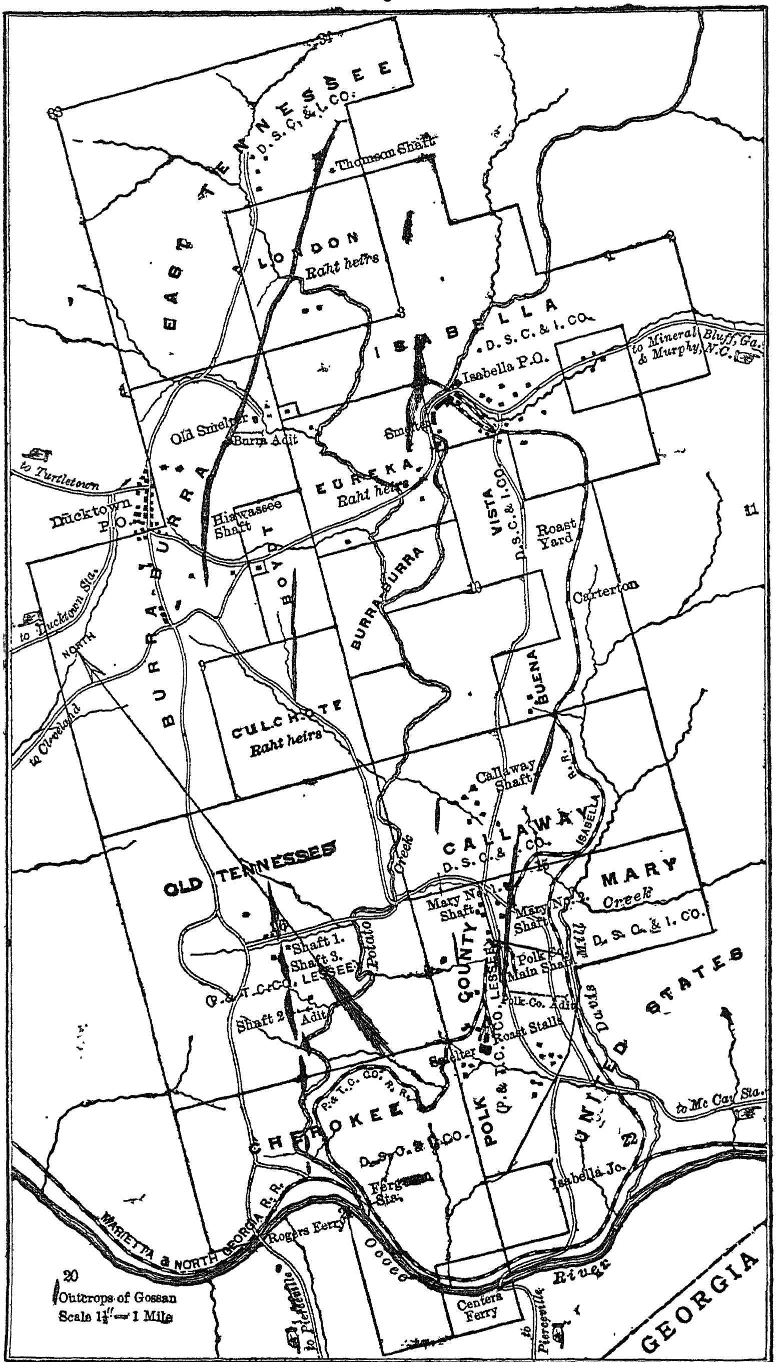
melted the richest carbonate ores, occurring in an easily-melting, self-fluxing, ferruginous gangue, in small crucibles made of a peculiar refractory but easily-cut rock of the neighborhood, and used for this purpose a furnace built of oblong rectangular brick cut from the same kind of rock as the crucible. This is the surmise of the writer from the relics of the industry seen by him.

As Judge Parks is in correspondence with the Smithsonian Institution with regard to this matter, it is to be hoped that a thorough search of the spot by excavation will secure, before it is too late, additional and comprehensive relics of a pre-historic industry.

The country around Ducktown is a portion of the lands purchased by the United States government from the Cherokees. Ducktown derives its name from a Cherokee village near the mouth of Davis Mill creek and near the present Isabella station on the Marietta and North Georgia Railroad; the name of the chief of that particular village being "Duck," the district was called Ducktown. In the same way, the adjoining district of Turtletown, to the north, received its name from the Indian chief "Turtle."

The country around Ducktown being densely wooded, and the soil very light and of the poorest kind, the mountainous parts of Polk county especially were settled at first very slowly and sparsely. Judge Parks, Senior, says that only a few years before the discovery of the Ducktown mines the entire tax-list of that region, which it was his duty at that time to collect, amounted to less than seven dollars. Money was then (about 1850) almost an unknown article in that community. People manufactured everything they consumed, and the small amount of commerce carried on was wholly barter. It was in the year 1849 or 1850 that copper-ores were first discovered in the Ducktown district. Chalcopyrite impregnated in the quartz croppings of the Hiwassee ore-body (see map, Fig. 2) in the bottom of a small creek southeast of and close to the present town of Ducktown misled the original discoverer to believe he had found a gold-mine. The sinking of a shallow shaft on these croppings in 1850 led to the discovery of the rich zone of partly oxidized and concentrated copper-ores—locally named "black copper"—which afterwards made Ducktown famous for its enormous de-

Fig. 2.



MAP OF THE DUCKTOWN MINES.

posits of rich copper-ores. Notwithstanding the intense mining excitement existing at that time, as a result of the California gold fever, the sparsely settled state of this district, together with its inaccessibility, delayed for more than a year the spread of the fame of the Ducktown copper-region. But at the end of 1852, and down to the middle of 1853, the discoveries of enormous and enormously rich masses of copper-ores engendered a regular mining furore. Any and every reddish or brownish coloring of rocks and every quartz-cropping in the region was made the basis of a mining venture, or, at least, of a mining company. Old caved-in shaft-holes and drifts run into the hills (mostly, however, for short distances only) still testify to the prevailing speculative activity of that time.

The more important and enduring mines or mining companies of that first period of the Ducktown mines came into existence in the following order:

Hiwassee,	August, 1850.
Cocheco,	October, 1850.
Tennessee,	October, 1851.
Polk County,	November, 1852.
Cherokee,	December, 1852.
Eureka,	April, 1853.
East Tennessee,	June, 1853.
Isabella,	July, 1853.
London,	September, 1853.
Mary,	September, 1853.
Callaway,	November, 1853.
Culchote,	February, 1854.
United States,*	August, 1854.
Biggs-Boyd,	August, 1854.

This list clearly indicates the middle of the year 1853 as the culmination of the early Ducktown mining excitement. This is easily accounted for by the recorded production of 808 tons of ore, averaging 28 per cent. of copper, by seven mines of the region, during the month of September, 1853. The recorded production by eight mines up to that date, since the discovery of copper, three years before, had reached 14,291 tons of rich copper-ore; *i.e.*, that amount of copper-ore had been shipped away from Ducktown, mostly to Swansea, Wales. The amount of ore shipped was then valued at over a million dollars.

* A "wild-cat" or non-producing mine, but owning a tract of about 1000 acres.

At the end of 1853, or beginning of 1854, smelting-works were erected at different places in the district. The remnants of some of these works still exist. The Burra-Burra Company, owning the Hiwassee and Cocheco mines, has the best-preserved of these old smelting-works standing on their property to-day. It contained four blast-furnaces of the German *Brillen-ofen* type, with two tuyeres to each furnace at its back and an exterior crucible for the separation of matte and slag.

The Union Consolidated Company, which controlled the Cherokee, Mary, Callaway, Isabella and East Tennessee mines, and also at times the Polk County and Tennessee mines, erected at first smelting-works on the Old Tennessee mine land, which belongs to the school-fund. Later on, smelting- and refining-works were erected on the land of the Isabella mine, where the village and post-office of Isabella is now located. These last-named works were kept running longest.

The Polk County Mining Company, owning the mine from which it derived its name, erected smelting and refining-works of its own on its lands. These works closed down, when the company failed, after exhausting the richest part of the black-ore zone of its mine. Subsequently, the property reverted to one of the original owners, after a protracted lawsuit.

Ducktown seems to have had its due share of the curse of every prosperous mining locality—lawsuits.

The Tennessee, or Old Tennessee mine, located on a section of land (No. 16) belonging to the school-fund, was leased to a company by the school commissioners. It contributed a splendid yearly income to the public school-fund. But some legal flaw of the lease executed by the school commissioners led to a lengthy lawsuit, and the stoppage of regular mining on the school-property. All revenue accruing to the school-fund from that source ceased. Then came in succession the robbing of the remaining black-copper deposits in the mine by "tributers," without any benefit to the school-fund, and an enormous claim for lawyers' fees against that section of land, standing against it to this day, and making it valueless to the school-fund itself. The value of the Tennessee mine was given during this celebrated lawsuit at from $200,000 to $1,000,000, based on its supposed possible output of copper-ore. Such values put on lands around Ducktown in sober, good faith by reputable

witnesses under oath, show better than anything else the exaggerated estimate then entertained of the inexhaustible resources of rich ores, which the mines were supposed to contain.

But already in 1860, at the beginning of the war of the Rebellion, the production of the rich black-copper ores had fallen off considerably. Experiments were in progress, under the energetic and competent management of Capt. J. E. Raht, to utilize the inexhaustible supply of low-grade sulphuret-ores—the so-called "yellow" ores—of the district in the production of copper.

The Civil War, of course, did not contribute to the prosperity of the mines. While none of the regular armies, either of the North or of the South, ever came into the vicinity of Ducktown, a guerilla warfare of the worst and meanest kind gave ample opportunity for the commission of atrocious crimes and outrages, in pursuit of private hatred and malice, of old family feuds, and most often simply for the gratification of a savage lust for plunder, robbery, cruelty, and murder. Even after the war, the peculiar location of Ducktown in the vicinity of almost inaccessible mountain-fastnesses, offering concealment from justice, and at the junction of three States, facilitating escape from the jurisdiction of either, made this region an ideal one for the law-breaker and "moonshiner." While law and order have been long restored in the community, the manufacture and sale of "moonshine" is still one of the leading local industries.

During the war, however, the Ducktown copper-production was a valuable and necessary industry to the Southern States; the copper produced by these mines was sorely needed for the manufacture of cartridges and other war material; and the mines, therefore, were not allowed to remain idle. At the close of the war most of the black-copper ore-deposits had become exhausted, and the sulphuret ores, of which an inexhaustible supply seemed in sight, proved too low in copper to pay for extraction. Experiments in concentrating the more slaty and quartzy sulphuret-ores were of as little avail on the whole, as experiments of leaching and precipitating the copper of the sulphuret ores, after a preliminary roasting. The fuel used at that time was wood for roasting, and charcoal for smelting in the small blast-furnaces, which were able only to treat a few tons of ore

per day. The dense forest originally covering the whole region had disappeared completely for miles around Ducktown; and the wood for the charcoal manufacture had to be brought from a distance of 10 to 20 miles. All supplies for the mines and works had to be hauled by teams 40 miles from Cleveland, Tenn., the nearest railroad point on the East Tennessee, Virginia, and Georgia Railroad. Under all these adverse circumstances, Capt. J. E. Raht managed to keep the mines going. But the company dispensed with his services, and instituted a monster lawsuit against the former manager, out of which Capt. Raht came with a complete vindication of his character and management.

The mines under new managers, unfamiliar with the details of the business (which only the closest knowledge of all details, coupled with an extraordinary ability of management, could keep alive), very soon became overburdened with debt, and got into the hands of a receiver. Mr. E. Miller, one of Capt. Raht's former assistants, was appointed by the court to this office, and before closing down the mines and works, succeeded in paying nearly all the indebtedness, by smelting the accumulated reserves of roasted ores. No new mining operations were allowed by the court, and in 1878 or 1879, when the last of the roast-piles had been put through the furnace, the smelting-works were closed down, and the first period of the Ducktown mines as copper-producers came to an end.

After this came a period of complete inactivity in mining, followed by starvation of the people remaining in the neighborhood, and the emigration of the large majority of a once large and thrifty population. Where in 1860 and 1870 probably four or five thousand people had found ample employment, in the years 1887 and 1888 probably less than five hundred tried very hard to eke out a scanty subsistence, hoping for better times from a revival of the once flourishing copper-mines.

The revival of the mining industry of Ducktown came through the building of the Marietta and North Georgia Railroad, connecting the city of Knoxville, Tenn., with the town of Marietta, Ga., which is only 20 miles north of Atlanta. This railroad gave to the Ducktown mines easy access to the coal-fields of Tennessee and Kentucky. Instead of charcoal at 10 cents a bushel (20 pounds), coke of a quality equal to Connellsville

could be laid down at Ducktown at a little above $3.00 per ton. It was also much less expensive to get machinery and supplies to the mines, and the product of the mines and reduction-works to a market.

Metallurgical art had also advanced wonderfully during that decade of idleness at Ducktown. When the mines and works shut down in 1879, the ores had been smelted in small old-fashioned blast-furnaces, built of brick and fire-proof stone brought from North Carolina. The area of these furnaces at the tuyere-level had been less than 4 square feet. The blast had been furnished through two tuyeres by an upright double-cylinder blowing-engine, driven mostly by water-power from over-shot wheels. These furnaces smelted probably 4 to 5 tons of charge each in 24 hours. Each required nearly as many men to run it as one of the modern water-jacketed furnaces, taking 100 tons of smelting-charge per day, which have taken their places. These large furnaces not only materially decrease the labor-expense per ton of ore for smelting, but they also use much less fuel per ton of ore. They allow longer smelting-campaigns, and require much less repair than the old ones. In fact, during that decade metallurgy had caught the spirit of our modern times, and metallurgical operations were carried on upon a scale probably beyond any dreams of the old metallurgists.

In mining also, much progress had been made. Dynamite had taken the place of black powder. The power-drill had to a large extent superseded hand-drilling. Opening a mine for a large production had become the work of months, instead of years. By working mines on a large scale and treating large quantities of ores, it had become possible to take into serious consideration the working of mines containing ores of lower grade than could be worked profitably in the old small way.

Many metallurgists and mining engineers of good repute had visited and investigated the Ducktown ore-deposits during the decade of their idleness. All of these investigators had found the ore too poor to pay if worked *for copper alone;* and as a sulphur-ore the pyrrhotite of Ducktown was justly condemned by these experts.* The fate of the district looked dark indeed.

* Arthur Wendt, *op. cit.*, says: "All of these (Ducktown) mines, if worked for

In 1889, when the M. & N. Ga. R.R. had been finished, the formerly productive mines around Ducktown were owned by the following parties:

The Union Consolidated Company or its successors owned the East Tennessee, Isabella, Callaway, Mary, and the Cherokee mines, besides much other land, which might have the name *mine* attached to it, but which contained neither known ore-bodies nor gossan-outcrops.

The Burra-Burra Company, which has since been reorganized, owned the Hiwassee and Cocheco mines, besides other land.

The Raht heirs owned the London, Eureka and Culchote mines, besides much timber- and farming-land in the neighborhood.

The Keith heirs owned the Polk County mine.

The Byrd Boyd mine was owned by successors to Boyd, the original owner.

The School Trustees had the power to lease but not to sell the Tennessee mine.

In 1889, an English company, the Ducktown Sulphur Copper and Iron Company, bought the interests of the Union Consolidated Company, or of its successors. They commenced operations in 1890 and spent considerable money in experimenting with a combined roasting-, leaching- and smelting-process, intending, as reported, to save everything contained in the ore—sulphur, copper and iron. A Mr. Guilstrap conducted the operations for about a year; but the process employed was evidently not adapted to the ore. The company then put up smelting-works with a modern Herreshoff furnace, capable of smelting over 100 tons of charge per day. They also put their Mary mine into suitable shape for the production of that amount of ore, and more. Since then mining and smelting operations have been carried on regularly and with apparent success by that company. It has also reconstructed the old narrow-gauge railroad of the Union Consolidated Company,

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copper only, have ceased to be valuable, and are not likely in the future to again become profitable as a source of copper *alone.*"

E. D. Peters, Jr., had condemned the Ducktown mines as unprofitable copper-mines, because of too low percentage in copper, and of no value as sulphur-ore, because the mineral was pyrrhotite.

connecting the Isabella mine, where the smelting-works are located, with the Mary mine, and has prolonged the track down the valley of Davis Mill creek (see Fig. 2) to a connection with the M. & N. Ga. R.R. at Isabella Junction. Moreover a third rail has been laid all the way from Isabella Junction to the Mary mine and on to the smelting-works at Isabella, where quite a village, with a post-office, has been established, and on to the Isabella mine.

Prior to the late financial panic in 1893 and the consequent drop in the price of iron-ore, another company, the London Iron and Coal Company had secured a lease from the Ducktown S. C. & I. Co., on the gossan-outcrop of the Isabella mine, and had mined and shipped a pure limonite-ore from that mine to iron-furnaces in Tennessee and Virginia. While somewhat high in silica and sulphur, this iron-ore is very low in phosphorus, and thus formed a welcome addition to the mixtures at southern blast-furnaces. The present low prices for iron-ores, however, have put an end, temporarily at least, to the mining and shipping of the Ducktown gossan-ores.

About a year ago the Ducktown S. C. & I. Co. increased the capacity of its works, which are now running regularly two large Herreshoff furnaces, turning out in one smelting operation a 50 per cent. matte from a roasted ore carrying probably not much above 3½ per cent. of copper. They treat about 200 tons of ore each 24 hours. A large roasting yard has been established between the Mary mine and the Isabella smelting-works. Here the ores are roasted in open heaps, protected from the weather by rough shed-roofs.

The Burra-Burra Company has done, during late years, considerable opening and prospecting work on the former Cocheco, now generally known as the Burra-Burra mine. An adit about 500 feet long has been driven from the lowest available point on their property across the east or hanging-wall of the ore-deposit to the vein, and after the vein had been tapped at about the level of the former black-copper zones, a level has been run south along the east wall for about 100 feet, gradually attaining greater depth below the old workings. But work has been suspended for some time by this company.

Some Pittsburgh, Pa., parties composing the Pittsburgh and Tennessee Copper Company, obtained in 1891, from the School-

fund trustees, a long mining-lease on the Old Tennessee mine. After doing considerable prospecting-work on the ore-deposits of that mine to various depths, down to more than 200 feet below the outcrops, the company, on the advice of the writer, who was at the time superintending its operations, secured a long lease of the Polk County mine from the owners of that property. In the summer of 1893 it began the work of opening that mine for a large production. In the spring of 1894 the erection of reduction-works was commenced after the plans of the writer, and the mine and works were connected by a narrow-gauge railroad with each other and with the Marietta and North Georgia Railroad at the mouth of Potato creek, at the new station of Ferguson, named after the President of the P. &. T. C. Co.

Unfortunately insufficient capital has so far prevented the completion of the works, according to the writer's plans; and the directors of the company have decided on a temporary suspension of operations.

Thus at the present time, the Ducktown S. C. & I. Co. is the only company carrying on active operations at Ducktown. The works of the Pittsburgh company are, however, so nearly completed, that there is hardly any chance of work remaining suspended there for any length of time, especially as the mine has been opened in a regular and systematic manner, and is fully prepared and equipped for a daily production of about 200 tons of ore, with the possibility of increasing this capacity. The quality of the immense ore-reserves proved to exist in the mine, leaves no doubt of profitable results with proper treatment and under competent management.

III.—General Geological Structure of the Ducktown Region and of its Ore-Deposits.

The geological age of the gneiss and micaceous schists, which mainly constitute the rocks encountered in the Ducktown basin, is as yet undetermined. They have been usually assigned to a formation known locally as the Ocoee slates or schists, which is prominently developed and splendidly exposed in the gorge of the Ocoee (and also in that of the Hiwassee) river, adjoining the Ducktown district on the west.

The Ducktown district has been included by the U. S. Geo-

logical Survey in that part of the Appalachian system in which the northwestern horizontal tangential force, uplifting the range, resulted in close-folding of the strata, *without faulting.** The truth of this conclusion, in a general sense, for the region surrounding Ducktown on all sides, the writer has had no opportunity to investigate. But, for the Ducktown district itself, this statement of close-folding, *without faulting*, is certainly not a correct description.

The lithological character of the Ducktown gneiss and micaceous slates, seems to differ materially from that of the semi-crystalline slates exposed in the Ocoee and Hiwassee gorges, in which, it is my impression, that the pre-eminently gneissoid character of the Ducktown rocks is wanting; and, in my opinion, a closer investigation of the rocks of the region would reveal the presence of two large faults—one along the eastern slope of the Frog mountains, and the other running along the west slope of Pack mountain, the steep face of which has a great resemblance to the west slope of Starr mountain, and others, along which large and prominent faults have been recognized.

Between these two faults in the Ducktown basin, geologically older rocks than either to the east or west would be exposed. This would probably account also for the presence of numerous smaller faults in the Ducktown region, in that geological province of the Appalachians where, according to the extensive researches of the U. S. geologists, generally close-folding of the strata *without faulting* exists.

Though not minutely acquainted with all the ore-deposits of the Ducktown region (but few of them having been opened to any decisive depths below their outcrops, and not all of those opened being now accessible), I feel fully justified, by minute observation of all those ore-deposits of the region with which I have become familiar, in maintaining *that the Ducktown ore-deposits are located along fault-fissures.*

The general geological structure of the Ducktown region, as I think it will probably be found on closer investigation, is depicted in Fig. 3, which represents an idealized section through the district in a northwest and southeast direction.

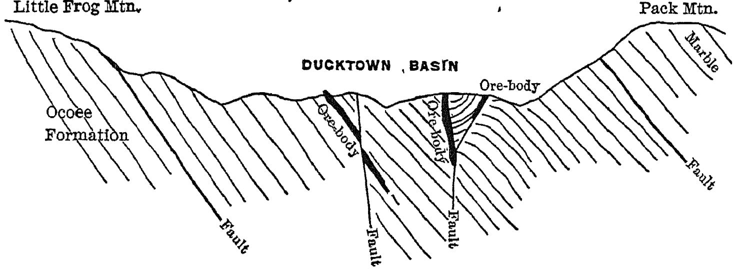
* See *Thirteenth Annual Report of the U. S. Geological Survey* 1891–92, Part II., p. 211, "The Mechanics of Appalachian Structure," by Bailey Willis.

The prevailing strike of the strata at Ducktown is parallel to the axis of the Appalachian range, and is, locally, about N. 20° to 25° E., approximately the same as that of the north and south lines of sectional land-surveys in that part of Tennessee. The prevailing dip of the strata is towards the southeast, generally near 50° to 55° from the horizontal.

To this prevailing strike and dip, however, there occur many exceptions, a closer study of which reveals that they are due to two causes.

The first of these is a bending or folding of the strata near a fault-fissure running parallel, or nearly parallel, to their strike,

Fig. 3.



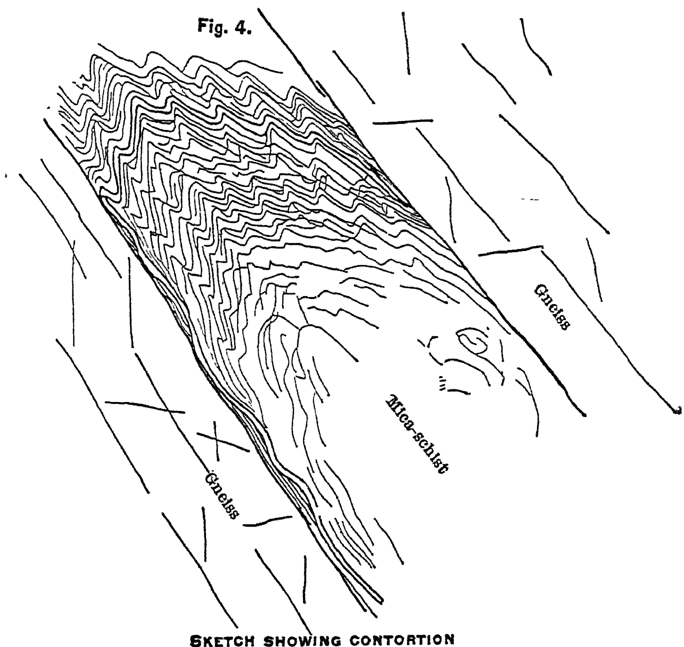
IDEAL SECTION N. W. TO S. E. SHOWING, PROBABLE STRUCTURE OF DUCKTOWN REGION.

i.e., in a general northeast and southwest direction. In dip, these fault-fissures, parallel to the prevailing strike, vary greatly. Frequently, they seem to have the same dip as the faulted strata, so that they would represent merely the sliding of one stratum on the other. When we examine the different layers of rock at Ducktown, we find them composed mostly of varying thicknesses of gneiss, alternating with highly-contorted, or minutely-folded micaceous schists. Frequently, we find a relatively thin layer of micaceous schist bedded between thicker and much harder layers of gneiss. In such cases, especially, we find the micaceous schist contorted in minute zig-zag layers and folds, which undoubtedly can be only the result of a sliding of the harder gneiss layers above and below the pliable layer of schist.

Such an occurrence, which can be found almost anywhere in the Ducktown district, I have sketched in Fig. 4, from an instance found on the north side of the road leading from Potato

creek to Copper hill, about 300 yards west of the west wall of the Mary ore-deposit, in section 15. The bedding-planes above and below this stratum of micaceous schist would, in reality, be fault-planes, as doubtless a dislocation occurred along these planes. In no other way, at least, can I account for the peculiar contortions exhibited in the mica schist. Probably, such dislocations along the bedding-planes of the strata were not of very large extent or "throw," and represent only the natural sliding along the more open bedding-planes of strata of dif-

Fig. 4.



SKETCH SHOWING CONTORTION AND ZIG-ZAG FOLDINGS OF MICA-SCHIST LAYER BETWEEN HARD GNEISS STRATA.

ferent degrees of hardness and pliability during the regional folding of Appalachian uplift.*

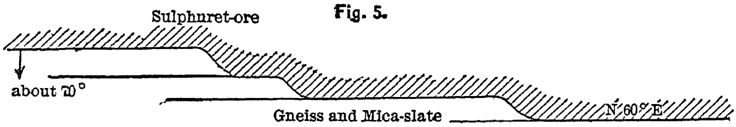
Other fault-fissures, which apparently have the dip of the strata, 50° to 55° southeasterly, do not follow the strike of

* That such foldings of the thinly laminated mica schists are produced by the bending and twisting which such a pliable rock naturally undergoes next to a fault-fissure, is shown clearly by their appearance next to the fault-fissure forming the east wall of the west ore-body in the Polk County mine, in the hard but thinly laminated gneissoid-rock, forming that east wall at the first level of that mine. See Fig. 22, after a sketch made in the mine. The foldings are much more regular and clear in nature than in my sketch.

the strata, but depart from it in their general course at an acute angle, usually somewhat further from the north and towards the east. While the general strike of the strata may be put at N. 25° E., these fault-fissures have a general course of N. 50° to 60° E. This general course of the fault-fissure at an acute angle to the stratification, is, however (frequently at least), the result of following a bedding-plane for some distance on the strike, and then jumping to another bedding-plane farther east, and so on.

Fig. 5 represents an occurrence of this kind along the east or hanging-wall of the Burra-Burra ore-body, south of the deep adit of that mine, and at the adit-level. This was the first

Fig. 5.



SKETCH SHOWING ROUNDED SMOOTH OFFSETS, OR BOSHES IN EAST (HANGING) WALL OF BURRA VEIN, AS SHOWN IN SOUTH DRIFT FROM DEEP ADIT, FEBRUARY 28, 1893.

clear proof which I encountered in the district, that at least one wall of the original sulphuret ore-bodies (in this case the east wall) was doubtless a fault-fissure. Sufficiently clear traces of slickensides and striae on this wall indicated plainly that, even after the formation of the ore-deposit, the movements along the fault-plane still continued. Such movements may be continuing at the present day.

Another proof that one of the walls of the ore-deposit was along a fault-fissure, was found in the old workings of the Polk County mine, represented in Figs. 6 and 7. Here, the old-level from an adit had been driven in the hanging-wall rock, and cross-cuts, longer or shorter, had been made at intervals across to the black-copper stopes. The sides of these cross-cuts furnished a good picture of the bending of the stratification of the country-rock as it approached the ore-deposit. This clearly revealed the boundary-plane of the ore-deposit as a fault-fissure; and the fact was afterwards amply verified by developments in the lower level of the same mine.

That movements took place at different times, and even in different directions, along the faults of this region, was clearly demonstrated in shaft No. 3 of the Old Tennessee mine, where

I had occasion to reopen an old cross-cut into the west or foot-

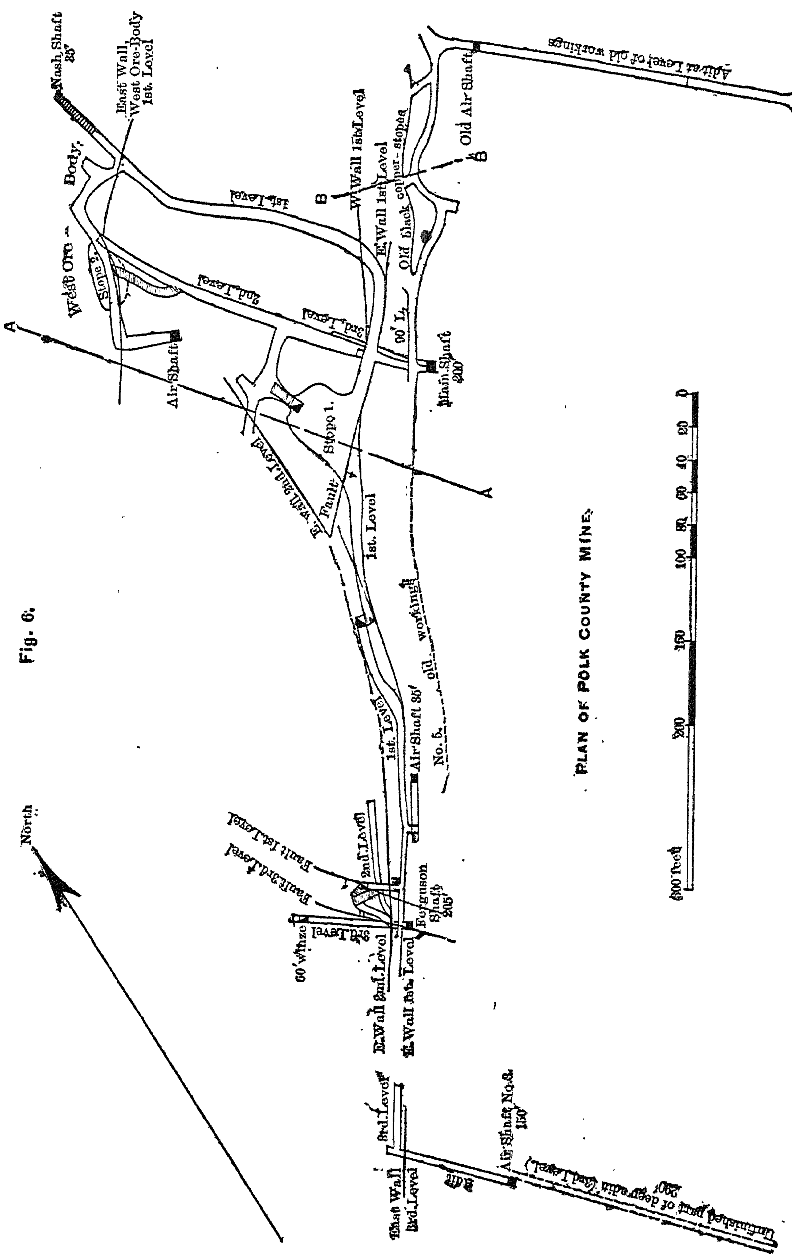


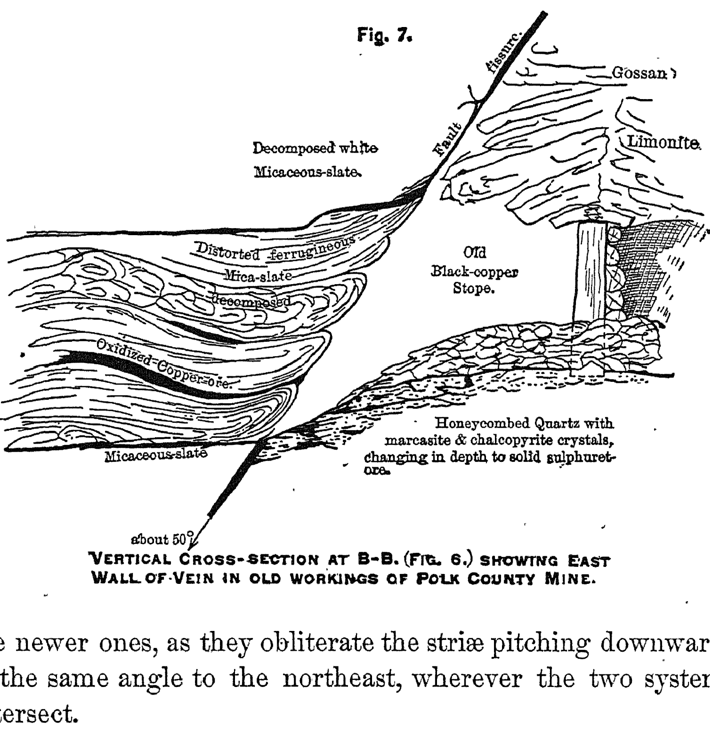
Fig. 6.

PLAN OF POLK COUNTY MINE.

wall of the ore-body, 100 feet below the surface. Here, the old miners had encountered a narrow fissure, 10 to 12 inches

wide, filled with a highly micaceous, clayey, soft, but dry substance, between walls of mica-schist. This fissure had been followed—parallel to the ore-body and to the general direction of the strata, but much deeper in dip—for a few feet on either side. It contained numerous slickensides and striæ, many of the latter crossing each other as shown in Figs. 8, 9, and 10; Fig. 8, giving a plan of the mine-workings; Fig. 9, the directions of the crossing striæ, and Fig. 10, the dip of the fissure with relation to the stratification. The striæ pitching downwards to the southwest at an angle of about 60° are evidently

Fig. 7.



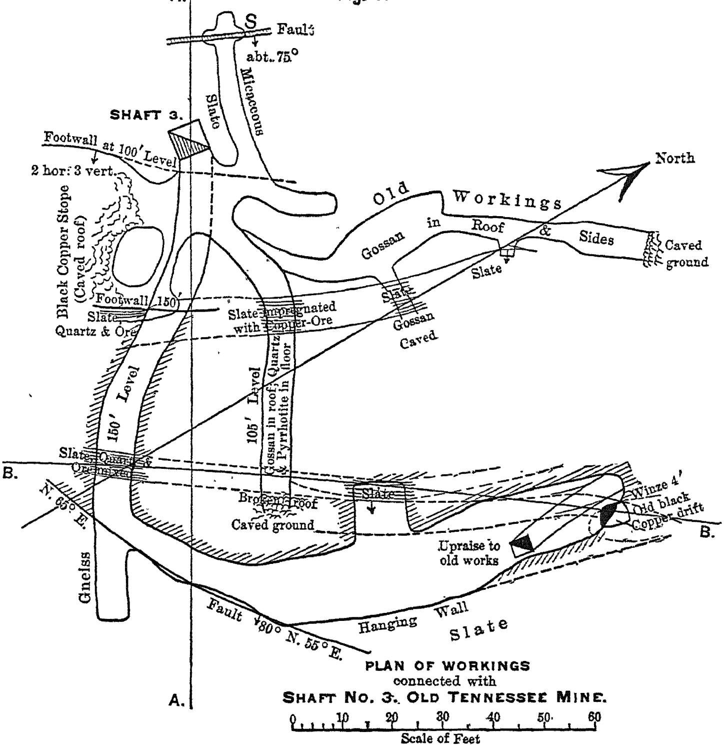
VERTICAL CROSS-SECTION AT B-B. (FIG. 6.) SHOWING EAST WALL OF VEIN IN OLD WORKINGS OF POLK COUNTY MINE.

the newer ones, as they obliterate the striæ pitching downwards at the same angle to the northeast, wherever the two systems intersect.

In sinking this same shaft, what is probably this same fault-fissure was struck about 155 feet below the surface. The drill-hole, penetrating to this fissure through its hanging-wall, filled with water directly, and the *ascending* current was so strong that, in the absence of pumping facilities, it had to be plugged to permit the driving of a cross-cut to the vein at that depth without interference from the influx of water. This jet, being good, pure drinking-water, had evidently no connection with any of the old flooded mine-workings, some of which were

below the water-level of that part of the country. This proved that some of the open fault-fissures in the country-rock (in this case in the west or foot-wall of the ore-deposit) were the channels of the *ascending* waters from the deeper regions of the country. A series of pretty large springs is found in a line parallel to the foot-wall of the Old Tennessee ore-deposit, and usually about 100 to 150 feet distant from it. In fact, several

Fig. 8.



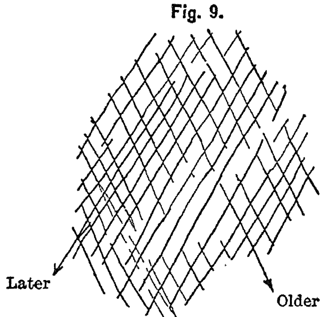
PLAN OF WORKINGS
connected with
SHAFT No. 3. OLD TENNESSEE MINE.

lines of such springs can be traced parallel to the different ore-deposits of the district in the west (usually the foot-wall) country-rock. Some of these, as, for instance, the main spring supplying the water for the Polk County smelting-works, come up through large fissures filled with white quartz. These fissures, running parallel to the fault-fissures or planes, alongside of which the ore-bodies have been deposited, and through which the ascending waters of the region now reach the surface as springs, are very likely also fault-fissures. The ore-deposits

themselves are very dry so far as opened in depth, no serious trouble ever having been experienced from any influx of water *inside* the ore-deposits or along their walls. All the water so far encountered in working these mines either has been surface-water, percolating through the old and mostly caved upper mine-workings, or has come from cutting water-bearing fissures in the country-rock of one or the other wall, mostly at some distance from the ore-deposit itself.

The fault-fissures along which the ore-deposits were formed have been filled completely and tightly with ore and vein-minerals, so that even where the vein-material is sharply and very clearly defined from the adjoining barren wall-rock (which is generally the case on the east wall), no open space is left which

Fig. 9.



SKETCH, SHOWING CROSSING OF STRIÆ OF SLICKENSIDES IN FAULT IN WEST WALL-ROCK OF OLD TENNESSEE VEIN, AT 100 FT. LEVEL, SHAFT 3. (S, FIG. 8) FEBRUARY, 7, 1893.

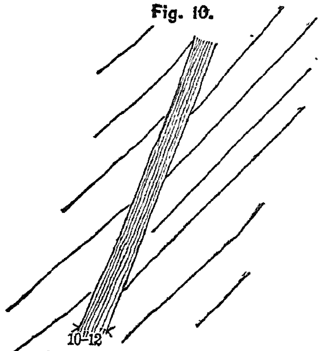
might serve as a channel for an ascending or descending current of the present water-circulation.

Fig. 11 is a sketch made by the writer of the northeast side of the collar of a caved shaft on the Boyd property. The bends of the different strata on either side of the vein-outcrop (which is very narrow at this shaft, as at many other places in the outcrops of the ore-bodies) show this body to lie along a fault-fissure. Not only are the strata flattened to nearly a horizontal position on either side of the fissure, but the ends of the strata are also bent upwards on the hanging-wall side and downwards on the foot-wall side of the gossan-body, thus affording unmistakable marks of a fault-fissure.

But besides these numerous fault-fissures parallel with the general course of the stratified country-rocks, or of the ore-

bodies themselves, we find in the strikes and dips of the country-rocks, away from the ore-deposits as well as in the underground developments of the mines, ample evidence of another system of foldings and faultings of the country-rocks, and also of the ore-deposits which they enclose. These point to a force exerted in a direction nearly at right angles to that which caused the general Appalachian uplift. The axes of these folds run W. of N.—E. of S., and the faults resulting from this tangential pressure exerted in a N.E. and S.W. direction on the rocks, strike N. 20° to 30° W. The dip of these faults, so far as they have come under my observation, is to the S.W., and mostly steeper than 65°.

Fig. 10.



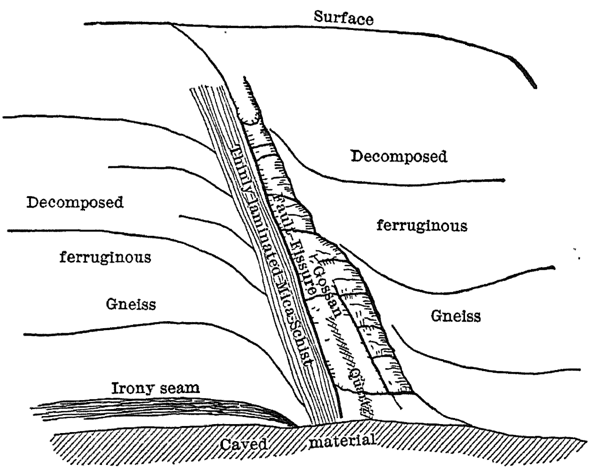
CROSS-SECTION (VERTICAL) OF FAULT-FISSURE IN WEST WALL-ROCK OF OLD TENNESSEE VEIN, (AT S. FIG. 8.) SHAFT 3; 100 FT. LEVEL.

Two folds or rolls of this system can be plainly observed to the north and south, respectively, of the center of the N. and S. boundary-line, dividing Sections 15 and 16. The one to the north shows plainly in the face of the steep bluff in the bend of Potato creek, on the east side, about 100 yards north of the county-bridge (see Fig. 17). The other may be traced in the outcrops of the hard gneiss and much-contorted mica-schists in the ravine south of the half-section corner, along the foot-path leading from the bridge to the Polk County mine. This latter roll or fold is probably a continuation of the anticline, along the axis of which the cross-cut on the second level of the Polk County mine has been driven from the main shaft to the ore-body.

Parallel to the axis of this anticline, a fault belonging to the

same system of cross-folding has been exposed in the Polk County mine near the Ferguson shaft. This fault, with finely developed slickensides and striæ on its walls, was first encountered in the winze connecting a cross-cut on the first level, about 25 feet north of the Ferguson shaft, with the intermediate second level driven from the same shaft. It is observable in the corner of the shaft, near the regular second level, and can be plainly followed from the shaft in the roof of the third level, in which the ore-body extends at least 25 feet further east on the south side than on the north side of the fault. The horizontal throw of this fault, which evidently occurred after the

Fig. 11.



SKETCH OF N. E. SIDE OF COLLAR OF CAVED BOYD SHAFT. STRIKE OF FISSURE: N 25 TO 30° E.; DIP EAST 70°

formation of the ore-body, is thus, at the third level, 200 feet below the surface, at least 25 feet to the east on its south or hanging-wall side. The dip is about 65° to 70° from the hori zontal.

The sinking of an inclined winze about 70 feet south of the main shaft of the Polk County mine from the first to the second level, and afterwards the opening out of stope No. 1 from that winze, disclosed a fault, or more probably two parallel faults, close together, as shown in Fig. 12, which is a vertical cross-section from N.W. to S.E., *i.e.*, at right angles through the general direction of the ore-body at that point.*

* Of course, the dotted lines on this section are mere guess-work, as it is im-

South of and opposite to shaft No. 3 of the Old Tennessee

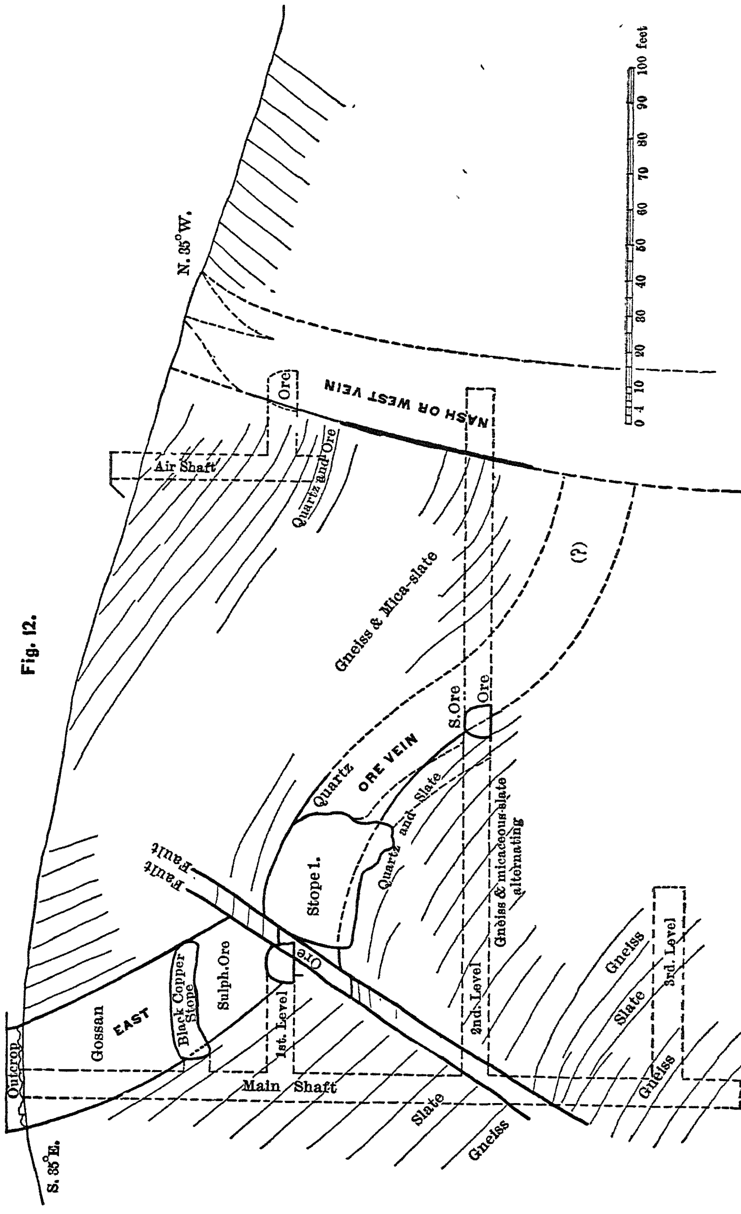


Fig. 12.

SECTION A-A. (FIG. 6.) POLK COUNTY MINE.

mine, a nearly vertical fault-fissure forms the east wall of the

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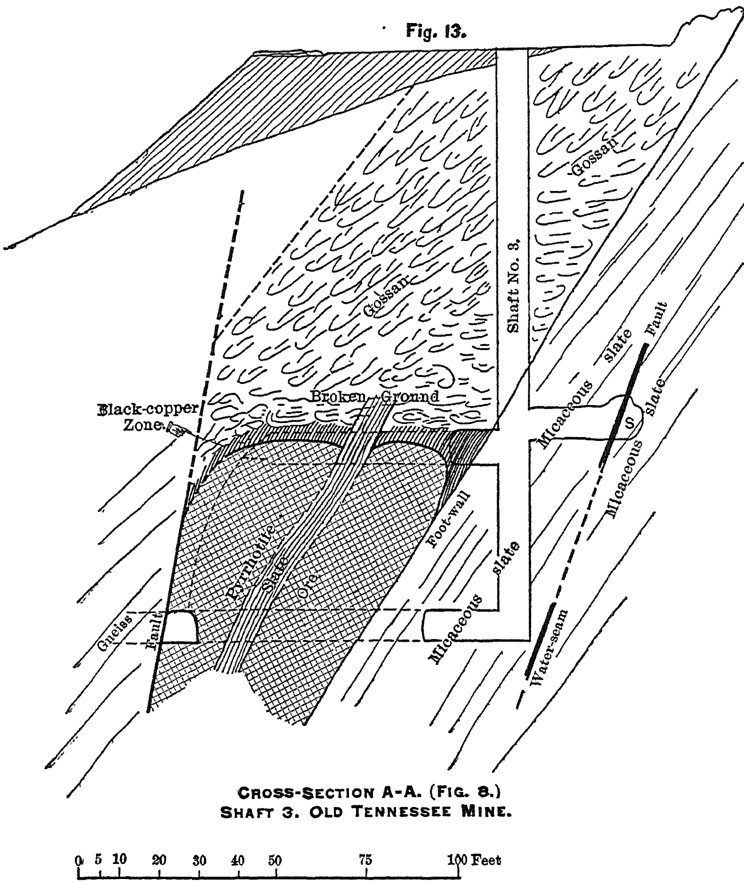
possible, on account of the local cross-roll or anticline in the second level, to draw conclusions as to the dimensions and course of the east ore-body.

ore-body on the 150-foot level, very probably up to the 100-foot level, and possibly to near the surface. This fault-fissure, where first encountered in the 150-foot cross-cut from shaft No. 3 (see Figs. 8, 13, and 14), had a strike of N. 65° E., and was nearly vertical, with a slight dip east; but when followed north it turned more to the north until, when the ore-body left it, its

Fig. 13.



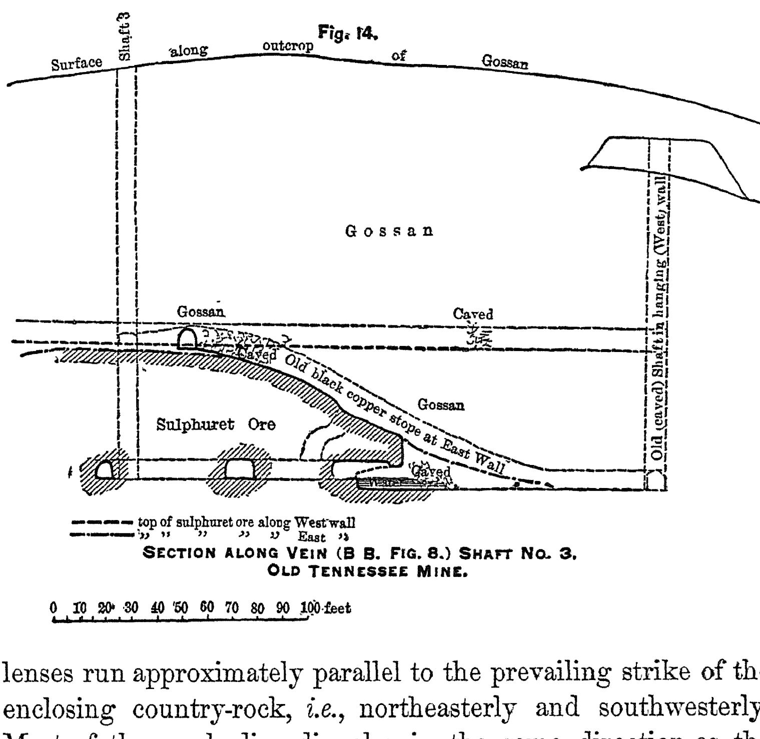
CROSS-SECTION A-A. (FIG. 8.)
SHAFT 3. OLD TENNESSEE MINE.

strike was N. 55° E. and its dip a little flatter. The foot-wall of the ore-deposit, tolerably well defined, had the usual strike of the country-rock, N. 25° E.; and if the foot-wall and the fault-fissure forming the hanging-wall kept their courses, the ore-deposit would speedily pinch out to the south, and would do so sooner at a lower level than at a higher one. Such a change could be traced at the outcrop; and the traditions of the old miners, who had been in the upper workings before

they caved, clearly pointed to the same conclusion for the ore-body in the old workings (the 100-foot level of shaft 3). According to the outcrop-indications, and to these reports, the southern continuation of the ore-body had been thrown horizontally to the west, by this fault, in the old workings, for a few feet more than the width of the body itself. Fig. 15 represents this horizontal throw as it probably appeared in the old black-copper workings south of shaft No. 3.

The Ducktown deposits present what would be usually called lenticular ore-bodies. The longer horizontal axes of these

Fig. 14.

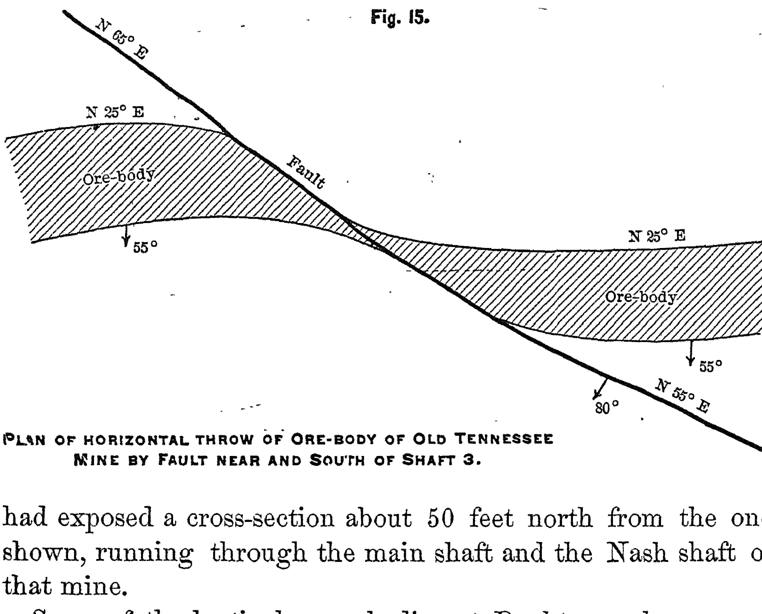


SECTION ALONG VEIN (B B. FIG. 8.) SHAFT NO. 3.
OLD TENNESSEE MINE.

lenses run approximately parallel to the prevailing strike of the enclosing country-rock, *i.e.*, northeasterly and southwesterly. Most of the ore-bodies dip also in the same direction as the country-rock, but usually at a steeper angle. They generally follow fault-fissures, nearly parallel to the stratification, but dipping steeply, as already explained. The horizontal, as well as the vertical lenticular, form of the individual, more or less disconnected, ore-bodies is largely due to the effect of fault-fissures cutting across the formation and ore-bodies at acute angles (and also at nearly right angles) and producing horizontal throws of from 30 to 50 feet, which, combined with the

thickness of the ore-deposits have resulted in very irregular and abruptly changing outlines of the horizontal sections of the latter. The opening of these ore-bodies for work on a large scale is likely to be full of surprises until actual developments have afforded a thorough knowledge of the local systems of faults and rolls affecting the particular deposit concerned. Not all the Ducktown ore-bodies, however, present such puzzling features as did the Polk County mine to the writer; and very few mines in the district will furnish such unusual cross-sections as the one depicted in Fig. 12. Probably this would have looked still more abnormal if the actual workings

Fig. 15.



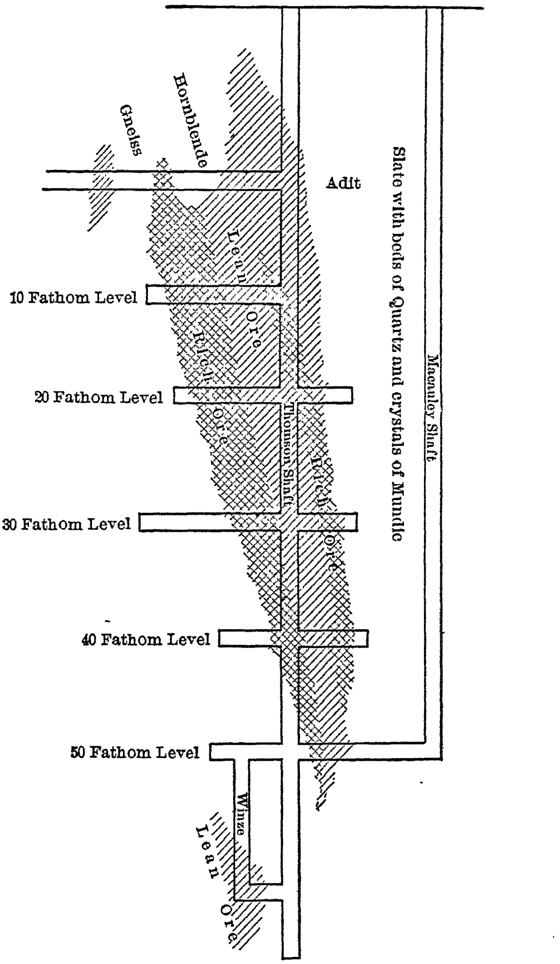
PLAN OF HORIZONTAL THROW OF ORE-BODY OF OLD TENNESSEE MINE BY FAULT NEAR AND SOUTH OF SHAFT 3.

had exposed a cross-section about 50 feet north from the one shown, running through the main shaft and the Nash shaft of that mine.

Some of the lenticular ore-bodies at Ducktown show a continuous uninterrupted outcrop of gossan thousands of feet in length. The width of the different ore-bodies varies from 12 feet to more than 400 feet. The former is shown at the narrowest point of an ore-body which the writer has ever encountered in the district, viz., opposite the main shaft in the eastern ore-body of the Polk County mine on the first level. The latter is the distance to which the diamond-drill is reported to have penetrated across the Isabella ore-body without finding the west wall of that deposit. Some of the Mary workings show a width much exceeding 100 feet; and the cross-cut on

the first level of the Polk County mine revealed a thickness of 75 feet from the east wall of the western ore-body to the west without reaching the barren country-rock, and this at a depth

Fig. 16.



CROSS SECTION OF THE EAST TENNESSEE MINE.

Scale of Feet

0 25 50 100 150 200 250 300

AM. BANK NOTE CO. N.Y.

of only about 45 feet below the outcrop of that ore-body. Yet a man can bestride that outcrop by placing a foot on either wall-rock.

The depth to which these deposits descend is as yet unknown.

The Mary mine has reached a working depth of 180 feet below its lowest adit, or about 300 feet below the collar of the main hoisting-shaft. At that depth the walls of the ore-body seem to be rapidly coming together. But there is every reason to believe that even if this upper ore-body should pinch out at not much more than 300 feet below the surface another ore-body would take its place. I have no doubt that one of the walls of this ore-body at the 30-fathom level (below the adit) is a fault-plane, cutting off and throwing the ore-body in a manner similar to that shown in Fig. 13, and that the continuation of the ore-body will be found on the other side of that fault-plane, possibly without a complete pinching-out of the ore—*i.e.*, possibly the two narrow ends of the severed parts of the ore-body may overlap.

At the East Tennessee mine, the out-cropping ore-body was completely worked out, during the first Ducktown mining-period, down to more than 400 feet below the outcrop; but, before mining was stopped, the top of a new ore-body had been found and opened up. The only representation known to me of the mine-workings and of the ore-bodies encountered in that mine is a vertical cross-section of the ore-bodies and main-shafts, given by Mr. Wendt in his essay already cited, and reproduced in Fig. 16. From this section no accurate conception of the actual condition can be formed. Apparently it has been compiled from different cross-sections (which ought to have been given to show the true state of affairs) in accordance with the compiler's preconceived theory of the occurrence of these detached ore-bodies, in depth as well as horizontally, *en echelon.*

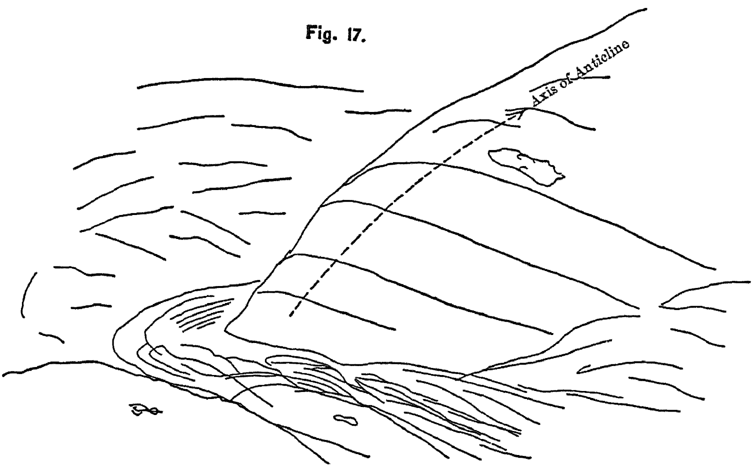
The different shafts and the winze shown in the cross-section are not in a line, and the picture gives, therefore, an erroneous impression. The writer is strongly of the opinion that if the workings could be investigated the ore-bodies would be found to be on the two sides of a fault-fissure, instead of *en echelon*, as shown.

At the Old Tennessee mine the writer has followed the ore-body for 230 feet vertically below its outcrop. At this depth it shows a thickness of at least 35 feet of solid sulphuret ore, with no indication of any near approach to the west wall. This was in shaft 2 of that mine.

In shaft 3, as Fig. 13 shows plainly, the fault-fissure forming the east wall must soon cut out the ore-body in depth opposite to that shaft; but the continuation of the ore-body would probably be found at a short distance below, on the east side of the fault-plane.

The Ferguson shaft of the Polk County mine has a depth of 210 feet below the outcrop of the ore-body. Four cross-cuts have been driven from it, through the east wall, in which the shaft is sunk for short distances to the ore-body and into or through it. The first level of this shaft is only a short distance below the old workings, and shows that part of the

Fig. 17.



BLUFF OF GNEISS MOSTLY ON EAST SIDE OF POTATO CREEK, SHOWING ANTICLINE AT RIGHT ANGLES TO PREVAILING STRIKE OF STRATA.

vein near the east wall, filled with solid sulphuret ore, to be about 28 feet thick.

The third level of the mine (*i.e.*, the fourth or 200-foot level of this particular shaft) shows a thickness of at least 75 feet of sulphuret ore at this shaft; and the west wall has not been reached. A winze, sunk during the first period of Ducktown mining, from the end of this cross-cut, at a point 75 feet west from the shaft, to a depth of 60 feet carried slaty ore all the way down, showing that the ore-body at this point reaches at least 260 feet below the outcrop; how much deeper is unknown.

The main shaft of the Polk County mine is down to 200 feet below the surface; but the third or 190-foot level, cross-cutting

from the shaft to the west, has not as yet reached the ore-body. Four diamond drill-holes, made under the direction of Capt. Raht, twenty or more years ago, left, however, no doubt that a body of copper-ore exists in that mine at a depth of 400 feet below the Nash shaft wider and richer than any so far encountered in the upper workings.

Whether the ore-body, which is 75 feet wide at 45 feet below the collar of the Nash shaft, will prove to reach to and below the depth of 400 feet, and the rich copper-ore revealed by the diamond drill will then belong to it, or whether these will prove to be two separate ore-bodies cannot now be predicted with certainty. The probability, however, is that the western ore-body which has been cross-cut by the first level of the Polk County mine, jointed and augmented by the eastern ore-body, or by a branch or spur from it, thrown to the west by the cross-anticline (followed in the second level cross-cut from the main shaft) will be found to extend continuously to a depth of 400 feet, where it will be much richer in copper and of much larger dimensions than at the first or second level of the mine. This, at least, is the writer's interpretation of the developments of the diamond-drill, viewed in the light of the experience gained in opening the upper levels of the mine.

To give an idea of the amount of ore contained in the Ducktown deposits, the Polk County mine (which is by no means one of the largest of these ore-bodies) will serve admirably. From what I positively know of this mine I feel justified in estimating the amount of ore extractable from the known bodies, to the depth of 400 feet (where, it must be borne in mind, the ore-body is larger than anywhere above), at 1000 by 40 by 250 feet, or 10,000,000 cubic feet, being very nearly 1,000,000 tons of ore.* This estimate leaves enough in the mine for pillars and support of the walls; and if rock-filling of the stopes is resorted to, as it ought to be, the total may easily be increased 50 to 100 per cent.

It is highly probable that the Ducktown ore-deposits extend to a greater depth than mining will ever be able to reach.

The Ducktown ore-deposits occur along three lines, parallel in a general way to each other and to the strike of the stratified

* This includes as ore all mixed ore and slate carrying more than 2 per cent. of copper.

country-rock. These lines or zones are about 1200 to 1300 yards apart, and are evidently located along three main fault-fissures, running nearly parallel with the stratification through the Ducktown district, as indicated in Fig. 3.

Besides these main fault-fissures, however, numerous smaller ones running in the same direction occur throughout the district; and consequently we find smaller outcrops of gossan scattered between the larger or main ore-bearing fissures. One of these smaller outcrops occurs in disconnected patches along a fissure running parallel to, and about 150 yards west of the Polk County and Mary ore-body. It is indicated on the map, Fig. 2.

All the large mines are north of Ocoee river and south of Stansbury mountain, which latter range of hills, as indicated above, is the most prominent result of the cross-folding exerted on the rocks of this region.

South of the Ocoee river much prospecting was done in the early days of the Ducktown excitement; and several small copper-mines were found and worked. The most prominent of these was the Mobile mine, the owners of which nearly worked out a body of rich ore to the depth of 150 feet, more or less, and had smelting works of their own near the mine. But all of the ore-deposits south of the Ocoee river are much smaller than the Ducktown ore-deposits proper. The ore-bodies are smaller and are fewer and farther apart. They seem to occur likewise along parallel lines or zones, apparently the continuations of those north of the Ocoee river; but the fault-planes, which are constant and deep-reaching north of the river, seem here to change gradually and for considerable distances to alternate with close folding of the strata, and to be much less prominently developed than in the Ducktown district. The ore-bodies south of the Ocoee have certainly much less chance of holding out either horizontally or in depth, besides being much smaller, as a rule, although I have seen stopes in the Mobile mine, 20 and 24 feet feet wide in places, which had doubtless once been filled with solid, or nearly solid, ore.

North of Stansbury mountain also gossan outcrops of smaller copper-ore deposits are reported. But what has been said of the deposits south of the Ocoee applies with probably still greater force to these.

IV.—Contents of the Ore-Deposits.

The Ducktown deposits are, as to their main metallic contents, bodies of pyrrhotite, with subordinate marcasite, pyrite, chalcopyrite, zinc-blende and galena. The portions near the surface, to a depth varying from 17 to 80 feet, have been changed into "gossan." Below this gossan, and above the unaltered sulphuret-ore is a zone of partly oxidized rich copper-ore, or "black copper." This zone was the object of the early mining operations at Ducktown. It contained rich copper-ores, distributed in patches of varying dimensions (but not as a continuous body) within a zone or layer extending more or less horizontally across the vein, in thickness varying from 2 to 8 feet, and more, above the sulphuret ore-bodies and below the gossan. Such a deposit would at times extend over the whole width of the ore-body, from wall to wall, and for long distances along the strike. If of such horizontal extent, it was also usually of considerable depth. In other places the "black copper" would be found only in small pockets or seams of limited extent, and then usually along one or the other of the walls of the ore-deposit. Detached deposits of such ores would also occur more in the center of the ore-deposit.

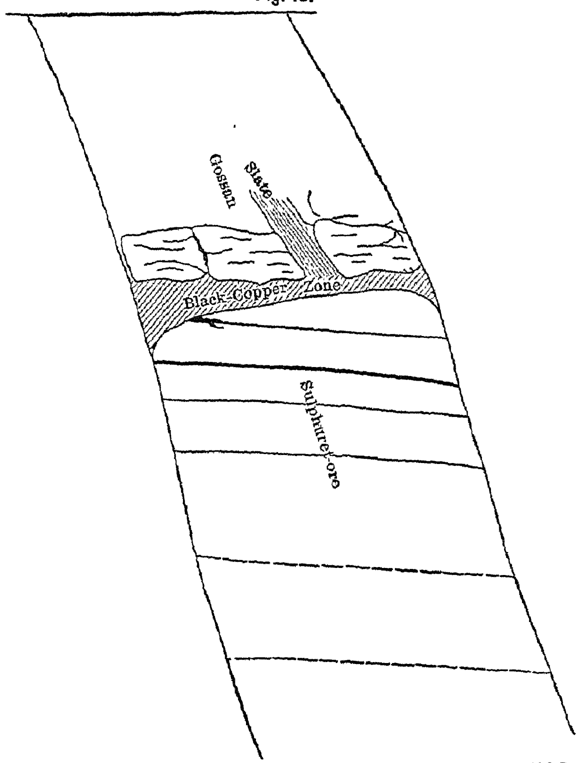
Frequently a floor of white quartz would be found underlying the black copper. This white quartz would frequently contain crystals and bunches of marcasite. In places the quartz is only sparsely impregnated with marcasite. In other places the quantity of marcasite in the quartz increases until the marcasite is the preponderating, and the quartz the subordinate constituent of the floor.

In the early mining days this floor of quartz or sulphuret ore underlying the more or less horizontal zone of black copper-ore would be called the "bed rock" by the miners. Thence originated a prevalent belief that the entire basin was more or less underlaid with a flat deposit of rich copper-ore. This belief gave rise to much useless prospecting. It, of course, soon gave way to a better recognition of the nature of the ore-deposits. But it was responsible for much over-estimation of the amount of *rich* copper-ore obtainable from the black copper zone of the district.

It is well to call attention here to a peculiar feature of these ore-deposits. As already observed, the lenticular bodies usu-

ally have a dip steeper than 50° from the horizontal, and ranging up to a nearly vertical position. A prominent characteristic of the unaltered sulphuret ore-bodies, more plainly observable in their upper portions near the black-copper zone, is the occurrence of smooth, nearly horizontal partings or floors extending through the body from one wall to the other, and for considerable distances in the direction of the strike. In the latter direc-

Fig. 18.



SKETCH, SHOWING POSITION OF FLOORS OR PARTINGS ACROSS ORE-BODY.

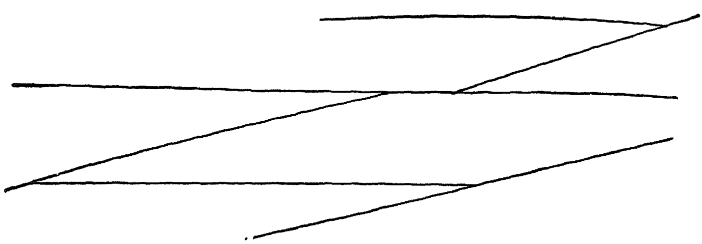
tion two adjoining floors not being perfectly horizontal may approach each other or may unite, and further on, may split again into two separate floors.

Figs. 18 and 19 show the occurrence of such partings across and along the ore-body, as observed in the workings of the Polk County mine and of the Old Tennessee ore-bodies. They are prominent also in the ore-body of the Mary mine, where they were supposed at one time, by the superintendent, to form

boundaries between richer and poorer ore—a view afterwards abandoned in the light of more extended experience.

All of these floors, or partings, which the writer had an opportunity of observing, in their extent *across* the ore-body, had a *slight* dip downwards in the same direction as the east wall of the deposit; *i.e.*, as if they might have been horizontal—in that direction, at least—at one time, when the ore-deposit had a flatter position than at present. The converging and diverging of these partings in the direction of the strike of the ore-body, however, makes any hypothesis of an original horizontal position of all of them untenable. The partings nearer to the black-copper zone are open in places to the extent

Fig. 19.



SKETCH, SHOWING POSITION OF FLOORS, OR PARTINGS, ALONG STRIKE OF ORE-BODY.
(Vertical Section)

of several inches, as if the lower part of the ore-body had shrunk and settled away from the upper part. In some of these open horizontal seams layers of white quartz are found, deposited, undoubtedly, after the formation of these floors and after they had opened.

By these observations, the writer has been led to the conclusion, that the pyrrhotite-deposits had been formed, and the floors or partings established in them, *before* the deposition of the quartz found in the vein, and also probably before the impregnation with marcasite of the white quartz of the floor of the black-copper zone.

This would indicate, as to the genesis of these ore-deposits, that the deposition of the pyrrhotite had long preceded the deposition of the quartz and of the marcasite found in the vein. The upper partings being of a much more open character than the deeper ones, when the quartz deposition took place, the

quartz had a chance to form sheets of considerable thickness, extending in these open partings from wall to wall. These flat sheets of quartz being impervious to the descending surface-waters, the decomposition of the original pyrrhotite-body into gossan by the oxidizing action of surface-waters would proceed downwards until it reached such a thicker coating or floor of quartz, which more or less effectually prevented further decomposition downward. That quartz sheet would then form the floor of the secondary deposition of rich copper-ore, leached out from the oxidized upper parts of the vein, and re-deposited, mostly as a mixture of copper-glance and oxidized copper-ores.

Until the writer found (in the first level north of the Ferguson shaft of the Polk County mine) such an open parting, with a sheet of quartz in it, at a distance of about 6 to 8 feet below the black-copper zone, the quartz floors of that zone, with their frequent impregnations of marcasite, forming in many places the floor of the old black-copper stopes, were a serious puzzle to him.

The gossan of the Ducktown ore-deposits is only in spots a limonite pure enough to be available as an iron-ore, without considerable previous culling. Much of the gossan contains too much micaceous slate or quartz to be valuable as an iron-ore, and masses and bunches of iron-ore alternate with masses and layers of slate and quartz. In places, however—as, for instance, at the Isabella and Eureka mines (located on the same ore-body), as also in places at the Old Tennessee and the Polk County mines, and probably elsewhere—the limonite furnishes a good iron-ore, the more valued as it is quite low in phosphorus—a quality not often found in southern iron-ores. As already stated, a large quantity of such gossan-ore was profitably mined and shipped from the Isabella outcrop before the late drop in the price of iron-ore.

The "black-copper" ore found below the gossan had very little real black copper-ore, or tenorite (CuO), in its composition. Most of the copper in it probably occurred as copper-glance, whence it derived its black color. Native copper and cuprite were not infrequently present in it in considerable quantity. The carbonates of copper, mostly malachite, and some silicate, were found usually near the edges of the black-copper ore-bodies and also in seams and stringers in the lower portion of

the gossan. The sulphate of copper (blue-stone), mixed with sulphate of iron (copperas), was by no means a rare occurrence. A body of this kind of ore, two feet thick, was found by the writer in the old workings of the Polk County mine, where it had been left by the old miners, probably as not being rich enough in copper for mining at that time. This body of iron and copper sulphate carried, in its crystallized state, about 6 per cent. of copper.

The decomposed wall-rock at the level of the black-copper zone is frequently impregnated, in seams and fissures, with the green carbonate of copper for a considerable distance (even 10 to 12 feet) from the vein.

The two following analyses of black-copper ore, taken from Mr. Wendt's paper, already cited, will serve to show the nature of that ore. This is now only of historic interest, as, practically, the entire black-copper zone of the district has been exhausted. There may exist in some of the old workings small pockets, overlooked in former times or too poor in copper to be then considered valuable. But it will not pay either to hunt for these remnants of former rich ore-bodies or to reopen the old, mostly caved workings for the mining of such remnants.

Analyses of "Black-Copper" Ore.

(Analyst, Dr. A. Trippel.)

	I. Per cent.	II. Per cent.
Oxide of copper,	5.75	3.80
Sesquioxide of iron,	1.50	.63
Sulphur,	18.75	25.40
Copper,	71.91	41.00
Iron,	.93	26.56
Soluble sulphates of iron and copper,	.72	1.78
	99.56	99.17

The contents of the unaltered sulphuret ore-bodies below the black-copper zone deposits are of most interest, not only from an economical standpoint, but also from the fact that this part of the ore-deposit represents the deposit in its original, unaltered state—at least, so far as it is possible to speak of *original and unaltered* as applicable to any rock-formation.

The character of the sulphuret ores is not uniform, either in the different ore-bodies or even in one and the same ore-body. In some cases—as, for instance, in the East Tennessee mine—

the ore occurs mostly impregnated in seams of micaceous and hornblendic slate. Sometimes the ore filling these seams will be of the thickness of a sheet of paper. Again, these seams of ore will enlarge to thick masses and solid bodies, which, in turn, may enclose sheets or lenticular bodies of slate.

A. F. Wendt, in his work already cited, gives, as an average of eight months' output of ore from the poorer portions of the East Tennessee ore-body, the following analysis:

	Per cent.
Copper,	5.5
Iron,	20.0
Sulphur,	30.0
Silica,	42.0

In place of silica, we should probably read slate or insoluble residue; for the East Tennessee ores were pre-eminently slaty, not quartzose ores. Moreover, the sulphur in the above analysis appears too high; for even if all the iron had been present in the form of pyrite or bisulphide, instead of being, as is probable, mostly pyrrhotite or nearly monosulphide, there is still more sulphur contained in the above analysis than could combine with the iron and copper or with any zinc and lead which might have been present and might have supplied the missing percentage in the analysis. If the iron had been reported at 30 per cent. and the sulphur at 20 per cent., the analysis would have agreed better with the general character of the Ducktown ores. It shows, however, that of this *culled* ore, from which the poorer slate had been thrown out, still more than two-fifths of its weight, and therefore about five-ninths of its volume, was slate-rock.

The richer portions of the East Tennessee ore-body are reported by Mr. Wendt to have averaged over 7½ per cent. in copper; and the writer has been informed by old miners of the region, that for a long time during the early operations, the ore shipped from this mine to the smelting-works used to run from 11 to 17 per cent. of copper. This agrees with the record, that at first only the richest ores of the Ducktown ore-bodies were mined, for shipment to outside smelting-works; and that, as these abnormally rich copper-ores became scarcer, the neglected reserves of poorer ores gradually became the object of mining, for the supply of the local smelting-works.

The sulphuret-ores of other ore-bodies, as for instance, the Mary mine, consist mainly of pyrrhotite with impregnations or enclosures of chalcopyrite. The latter ore is the real copper-ore of the district, the pyrrhotite containing only traces of copper, and these probably only due to minute inclosed particles of chalcopyrite, overlooked in an examination by the naked eye. A superficial inspection would pronounce much of the sulphuret-ore of the Mary mine to be solid pyrrhotite with chalcopyrite; but closer examination always reveals the presence of other gangue-minerals, even in the most solid pieces of this pyrrhotite ore. The most prominent of these gangue-minerals is hornblende, probably, for the most part, actinolite and tremo-

Fig. 20.

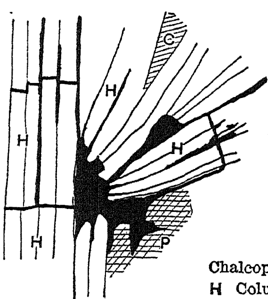
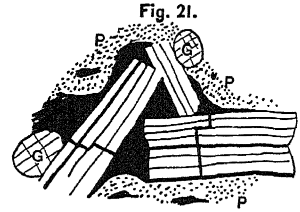


Fig. 21.



Ore from Polk County Mine.

Chalcopyrite Seams and Masses shown in heavy black;
H Columnar Hornblende Crystals; G Quartz Grains;
P Pyrrhotite; C Calcite.

lite. In fact, a characteristic feature of these ore-bodies is the presence of hornblende minerals in vastly greater quantity than in any other rock of the region. The hornblende crystals, where present in considerable quantity, occur usually in parallel slender columns, of a light greenish-gray color, often transparent. These bundles of columns cross each other at all angles. In the interstices between these crystals the chalcopyrite and pyrrhotite are deposited. The sides of the single columns of a bundle of parallel ones, are frequently coated with a thin film of chalcopyrite or pyrrhotite. Thin films of chalcopyrite also occupy some of the cleavage-planes across such a bundle of columnar hornblende.

Figs. 20 and 21 show on a somewhat enlarged scale this occurrence as frequently observed by the writer in the pyrrhotite of the Polk County mine. Quartz occurs, in subordinate quantity, in glassy, transparent, colorless grains as an admixture in the pyrrhotite. These quartz grains are sometimes deposited

on bundles of columnar hornblende, and when carefully detached, show the columnar impress of the hornblende on them, proving that this quartz was deposited after the hornblende crystals, and probably also after the chalcopyrite and pyrrhotite, which permeate the hornblende crystals, had been formed. This agrees with the already observed deposition of quartz (and of marcasite) in the horizontal partings of the ore-bodies. The quartz is doubtless a later addition to the ore-body.

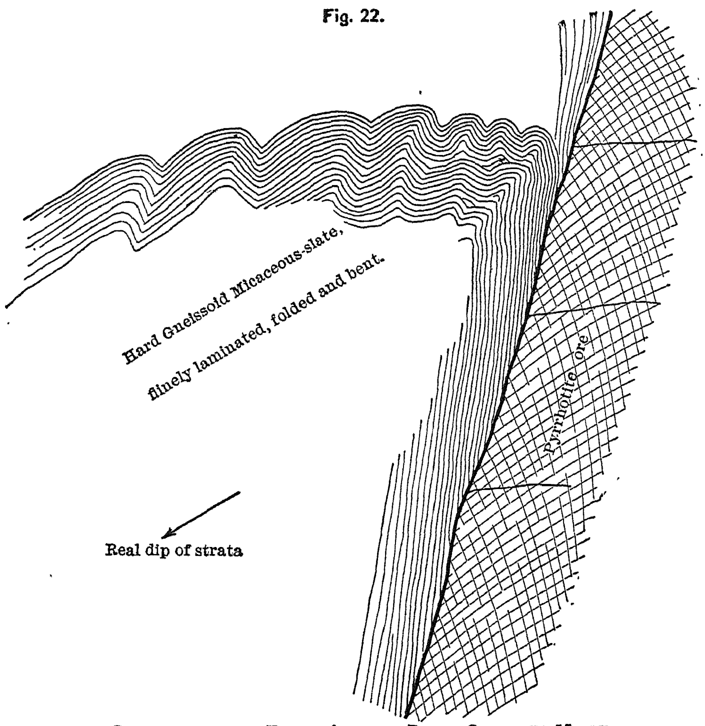
Among the rarer minerals found in the Ducktown ore-bodies are garnets, sometimes of fine yellow, orange-red and ruby-red colors, and often in clusters of beautiful regular crystals. I have not seen any of these garnet crystals, which were not accompanied in the immediate neighborhood by more or less chalcopyrite. But these garnets not only occur in juxtaposition to chalcopyrite and pyrrhotite; the crystals themselves are often completely permeated, or impregnated, with chalcopyrite and pyrrhotite. The writer was able to present to Mr. G. F. Becker, of the United States Geological Survey, a garnet crystal more than an inch in diameter, thus permeated by pyrrhotite and chalcopyrite. It was taken from shaft No. 2, of the Old Tennessee mine, from the center of the pyrrhotite ore-body. And I have before me, at this writing, two rhombododecahedrons of garnet, permeated throughout with pyrrhotite and chalcopyrite, which were taken from the ore hoisted out of the Mary mine.

This occurrence points clearly to the deposition of the pyrrhotite and chalcopyrite at the time, when these garnets were formed. Garnets being regarded as eminently the product of metamorphic action, it would seem that these pyrrhotite-deposits were originally deposited during a period of great metamorphic action on the rocks in which we find them. Considering also the amount of hornblende rock (abnormally large for this region) present in the ore-deposits, we might be led to inquire, whether this large amount of hornblende was not the product of metamorphic action on a rock containing much pyroxene, formerly occupying the place at present filled by the pyrrhotite ore-deposits. The hornblende would be the altered pyroxene of that rock, while its other component parts would have been replaced by pyrrhotite and chalcopyrite. There is certainly no rock outside of these ore-deposits existing

at the present time in this district, which would furnish sufficient material for the large amount of hornblende found in many parts of the deposits.

As I have said, hornblende crystals are nearly always present, even in the most solid pyrrhotite ore-specimens. The more solid pyrrhotite-ore is usually found towards the east walls of

Fig. 22.



SKETCH FROM FIRST LEVEL, POLK COUNTY MINE.

Showing foldings in hard Micaceous-slate adjoining East Wall of West Vein, Polk County Mine, and producing appearance of finely laminated layer of Mica-Schist dipping with the wall

the ore-deposits. But when (as for instance in the different crosscuts driven from three of the levels of the Ferguson shaft of the Polk County mine towards the west wall of the ore-body) we approach the west wall, we often find a rock composed almost completely of an aggregate of hornblende crystals. These hornblende crystals occur in columnar bundles crossing at all angles, where the rock is coarsely crystalline, and in radiating bunches of fine needles, where the rock is finely crystalline, or more compact. This rock is still a part of the ore-

deposit, containing more or less chalcopyrite (and pyrrhotite) impregnated in the interstices and seams of the hornblende crystals. When these impregnations of chalcopyrite predominate, the rock forms frequently the richest ore in the veins. No such hornblendic rock is met with anywhere in the district, outside of these ore-deposits.

Large masses of white quartz are often found in the veins, usually at one or the other wall. When they occur nearer the center of the deposit, these quartz masses will be found on closer investigation to extend to one of the walls, and then probably to extend along a fault-plane cutting obliquely, or squarely, across the ore-body. Larger bunches or masses of quartz may, however, be encountered in the vein, just as we find the small grains of quartz permeating the pyrrhotite ore; but it is a significant fact that this quartz does not contain any pyrrhotite or chalcopyrite, but contains often pyrite and more frequently marcasite.

Another ingredient of the ore is calcite. This carbonate of lime is found in some of the ore-bodies in surprising quantity, when we consider the total absence of limestone from the district for many miles in any direction and the total absence of any lime-salts from any of the waters flowing from the numerous springs of the region. There is not a sign of any scale in a boiler, after using spring-water in it in Ducktown day and night for months. That under these circumstances, carbonate of lime should be a distinguishing ingredient of an ore-body, to such an extent, that the average contents in lime of about 1500 tons of ore, taken without culling from the stopes of the Polk County mine between the first and second levels (70 to 125 feet below the outcrop) and taken from both ore-bodies, should be from 5 to 6 per cent., indicating at least 10 per cent. of calcite, is certainly proof sufficient, that this calcite must have come either from considerable depths, where rocks exist, which can furnish more lime than the gneisses and slates exposed above; or else, this 10 per cent. of calcite may be, like the hornblende of the ore-deposits, the altered remnant of the original (eruptive ?) pyroxene rock, now replaced by the pyrrhotite ore-body.

Detached masses of mica-schist, of more or less lenticular shape, and more or less parallel to the walls of the ore-bodies,

are often found within the ore-deposits. They contain flakes of mica, looking jet-black in the mine, but more or less dark brown in daylight. These masses of mica schist are usually barren, except when much contorted and thereby fissured and seamed. In these fissures and seams, which are never of great thickness or extent, stringers and veinlets of solid chalcopyrite and pyrrhotite are deposited. When these seams are numerous, and the filling (as is sometimes the case) consists mainly of chalcopyrite, these otherwise barren masses, or horses of dark mica schist furnish some of the richest copper-ore of the mine. This condition of affairs is easily recognized even underground, as the bright yellow of the chalcopyrite makes a strong contrast to the jet black of the mica schist. It is a peculiarity, well worth mentioning, that this dark brown mica schist is found *very rarely outside* the ore-bodies, the mica schist of the walls and of the country-rock being almost without exception of a light color.

The east wall of the ore-deposits is always sharply defined, and quite frequently, as I have said, is a fault-plane cutting obliquely across either the strike, or dip, or both, of the country-rock. Next this wall, at shaft No. 2 of the Old Tennessee mine, a rich streak several inches wide, and containing much solid chalcopyrite, was found in that ore-body, where shaft No. 2 penetrated the hanging (east) wall into the ore-body at a depth of about 110 to 125 feet below the surface. This rich streak was in strong contrast to the pyrrhotite ore adjoining it for a developed width of 30 feet, which contained an abnormally small amount of chalcopyrite. Near this wall, also, in the same mine near shaft No 3, was found an ore-shoot containing 2 to 3 per cent. of copper, which was considerably above the quality of the ore found in the other openings made under the writer's direction in the sulphuret ore-bodies of the mine.

In the Polk County mine, along the east wall of both ore-bodies, much galena, and especially much zinc-blende, occurs in the ore. This is particularly the case in the western ore-body between the Air shaft and the Nash shaft. In the first level driven from the Air shaft northwards towards the Nash shaft cross-cut along the east wall, frequently more zinc and lead appeared in the ore, than copper. Some zinc-blende and galena are also found in places more in the center of the ore-bodies;

but these zinc- and lead-ores follow pre-eminently the east wall of the ore-deposit, and when found more towards the center, occupy probably (like much of the quartz) fault-planes cutting across the ore-body. They are probably of later deposition than the pyrrhotite and chalcopyrite. Zinc-blende occurred also as a regular component of the East Tennessee ores, as the following analysis of the cinder from the smelting of the East Tennessee ore exclusively, during the years 1874 and 1875, will show.

	Per cent.
Silica,	37.20
Oxide of iron,	47.70
Alumina,	8.15
Lime,	2.52
Magnesia,	4.80
Oxide of manganese,	trace.
Oxide of zinc,	2.24
Sulphur,	.17
Copper,	.28
	103.06

This analysis is taken from Mr. Wendt's work. The excess in the total is probably due to the estimation of iron oxide as Fe_2O_3, instead of FeO, which it mostly is.

V.—Conclusions Regarding the Genesis of the Ore-Deposits.

Having been engaged for three years and a half in opening two of these deposits, located on different belts, and having had opportunity during that period to become more or less familiar with all the mines now accessible, I have been led by my observations (the more important of which have been recounted above) to the following conclusions as to the genesis of these ore-deposits:

When I arrived at Ducktown in 1891, to take charge of mining operations, I was prepared by previous reading to recognize in these deposits examples of regular *ore-beds*, as which they are cited by Groddeck.* Yet I had serious doubts whether such conditions ever existed in the terrestrial atmosphere, at a time when oceans were present on the earth's surface, as Groddeck conceives necessary to explain the deposition of *metallic*

* *Die Lehre von den Lagerstätten der Erze*, Leipzig, 1879, p. 115.

sulphides in regular *ore-beds* at the bottom of the sea. But I had a preconceived idea that I should probably find these ore-deposits to be true *bedded-veins;* my first general investigation of the district made in July, 1891, for the Pittsburgh and Tennessee Copper Company, confirmed me in this view, and I made my report in accordance therewith.

My views were subsequently changed upon closer and more detailed observation of the peculiarities of these ore-deposits. I would now suggest the following conclusions for investigation by competent observers, possessed of the time, opportunity, and necessary apparatus for thorough lithological examinations:

1. The Ducktown ore-deposits were deposited along fault-fissures, which cut through the stratified, metamorphic country-rocks at an acute angle to either the strike or the dip, or both.

2. Such a fault-plane always forms, so far as the writer has observed, the east wall of the ore-deposit.

3. This east wall is, or was originally, the hanging-wall of the deposit.

4. Where the east wall is now the foot-wall, it has been made so by the cross-folding and faulting of the rocks, in a direction approximately at right angles to the regional folding of the Appalachian range.

5. The east wall of the ore-deposit is always clearly defined; a sharp division existing along that wall between the barren country-rock and the ore-bearing vein-matter.

6. The same is probably true of the west wall of the ore-deposits; *i.e.*, it is true wherever the writer has had an opportunity to inspect that wall.*

7. The places of the present ore-deposits were originally occupied by (dikes of eruptive?) pyroxenic rock.

8. At the time of the uplift of the Appalachian range, or shortly subsequent (geologically speaking) to that time, these rocks were subjected to the metamorphic action of super-heated waters, ascending and depositing the pyrrhotite and chalcopyrite, and at the same time producing the metamorphic hornblende- and garnet-rocks now constituting the gangue of the

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* Cross-cuts towards the west wall of the ore-bodies have often been stopped on encountering the hornblendic rocks, or quartz rocks, which rocks really belong to the ore-deposit or vein, but have been mistaken for the west walls of the ore-deposits.

deposits; the other original constituents of the (eruptive) rock having been replaced by the pyrrhotite and chalcopyrite. The horses of dark micaceous schist found *within* these ore-deposits were masses of micaceous slate (wall-rock) detached from the walls, and more or less surrounded by the eruptive rock during its deposition along the fault-fissure.

9. The deposition of zinc-blende, galena, and, last of all, quartz, marcasite and pyrite, followed in later periods that of the pyrrhotite and chalcopyrite, and completed the transformation of the pyroxenic rock into the present ore-deposit.

10. Prior to the end of the period of the deposition of the quartz, however, must have taken place the cross-folding and faulting of the stratified rocks and of the enclosed ore-deposits, with, probably, the formation of the nearly horizontal parting or floors of the ore-deposits. This later disturbance of the rocks of the region, preceded a deposition of zinc-blende and galena, calcite (?) quartz and marcasite and pyrite in the faulted and folded ore-deposits, and probably along the newly-formed fault-fissures, which at least served as channels for ascending mineral-bearing solutions to the interior of the ore-bodies.

11. This later deposition of quartz probably went on until the channels referred to were choked by the process; this would have ended the formation of the sulphuret ore-bodies, as we see them at present (apart from the zones of surface-alteration).

12. The continuation of these ore-deposits in depth to any level at present within reach of human activity seems to be beyond reasonable doubt; and there seems to be no reason, nor any indication thus far encountered, which would point to a diminution in the copper-contents of the ore-bodies with greater depth.*

VI.—Mining.

The percentage of copper varies in the ores of the different ore-bodies, and of different parts of the same body. Considerable opening- and prospecting-work ought to be done ahead of the stoping, in these large ore-bodies, in order that the mines may be worked in a rational way, and the leaner portions of

* The deeper parts of the ore-bodies (leaving out of consideration the secondary concentration of the "black-copper" zone) are not the more barren. In the Polk County mine, as I have shown, they are richer in copper.

the ore-deposits may be used as pillars for the support of the walls. The pyrrhotite-ore is very hard; and, if judiciously worked, the mines will require very little timbering. Large faces may be worked in the stopes; and the ore, if extracted in a rational, systematic manner and on a large scale, may be mined and delivered above ground at a cost permitting the profitable utilization even of low grade material, as will be hereinafter more fully shown.

The proper method of working requires careful consideration for each particular mine. The best and most economical way of extracting the ore so as to obtain all or nearly all the *pay-ore* contained in the body, will depend largely on the question, whether all the ore of the deposit can be profitably mined and smelted, or whether some parts are too poor in copper to yield a profit. In the first case, some mining-method with a rock-filling of the immense open spaces made by the extraction of the entire ore-body would be indicated; while in the second case, the poorer portions of the ore-bodies should be utilized as pillars and supports for the walls.

VII.—Present Treatment of Ore by the Ducktown Sulphur, Copper and Iron Company.

Practically all the ore treated by this company (which I shall designate hereinafter for short as the Ducktown Company) is obtained from the Mary mine, and is in the main, a solid pyrrhotite-ore. But, as hoisted from the mine, it contains a considerable amount of slate-ore, *i.e.*, micaceous slate, and hornblendic rock impregnated with pyrrhotite and some chalcopyrite. The ore is crushed in Blake crushers at the mine, and then screened. The coarser crushed material goes over an endless picking-table, where a number of boys remove as much as possible of the slaty ore. The object of this process being simply to get the slate and hornblende out of the ore, so as to secure a basic ore for the subsequent smelting-process, some of the ore thus thrown out and put on the waste-dump, is richer in copper than the pyrrhotite-ore from which it is culled. The culled ore is then taken over the company's railroad, in cars holding 3½ to 4 tons of ore each, to the roast-yard at Carterton (see map, Fig. 2), where it is roasted in heaps under sheds. Each one of these heaps contains about 200 tons of crude ore. They are

arranged in parallel rows, with railway tracks between, and their ends abut upon the railway, which is at a little lower level than the foundation of the heaps. Wheelbarrows are employed for conveying the crude ore from the cars to the roast-heaps, and the roasted ore back into the cars. The roasting is done by contract, at so much per ton for all labor performed in unloading the crude ore from the cars, preparing the wood-foundation of the heap, putting the ore into the heap, firing and tending the same, and loading the roasted ore into the cars again. The fuel is furnished by the company; but the contractor is held responsible for the successfnl roasting; *i.e.*, he is only paid for the tonnage of successfully-roasted ore accepted and delivered to the smelting-works. Unroasted or poorly roasted ore contained in the burnt roast-heaps, and the heap-matte (or as much of these classes of material as he cannot, without detection or objection, mix with the well-roasted ore), he has to roast a second time.

The roasted ore is brought by railroad to the smelting-works at Isabella (see map, Fig. 2), and there dumped into the ore-bins. The bottom of these bins is on a level with the charging-floor of the smelting-works. The excess of "fines" which cannot be taken care of in open heap-roasting is roasted in shelf roasting-furnaces, of which the company has an abundance. They are a legacy of former experiments.

The roasted ore is smelted in one operation, in two large Herreshoff furnaces, with movable fore-hearths, into matte, containing, on an average, 50 per cent. of copper, which is shipped to eastern refining-works.

The heap-roasted ore seldom contains less than 7 per cent. of sulphur. In copper, it very rarely falls below 3 per cent., and seldom exceeds 4 per cent. Probably from 8 to 12 per cent., and more, of pure quartz is required to flux the roasted ore in the furnace. The silica of the resulting slag is above 30 per cent. To concentrate such an ore (containing 3 to 4 per cent. of copper and from 7 to 10 per cent. of sulphur), in one smelting, into a 50 per cent. copper-matte, requires, even in the Herreshoff furnace, which is excellently adapted to that kind of work, an abnormally high blast-pressure. The pressure used is never less than 14 ounces, and frequently above 16 ounces to the square inch. With a vertical distance of about 8 or 9

feet from the tuyere holes to the top of the charge in the furnace, such a high pressure naturally results in the production of a very large amount of flue-dust, especially with a roasted ore, handled several times before being put into the furnace, and containing a large amount of "fines." The resulting flue-dust is somewhat lower in copper than the roasted ore used in its production, and, not being considered valuable enough for re-treatment, is thrown over the slag-dump. The dust-chambers seem to be used solely to keep the dust from settling on the roofs of the smelting-works.

The loss of copper in the flue-dust is estimated by the writer at not less than from 2½ to 3 per cent. of the copper contained in the charge put into the furnace; but this quick concentration, in one smelting, of a 3.5 per cent. ore into a 50 per cent. matte, involves a much more serious loss of copper.

In making such an extreme concentration it is impossible to keep the slags low in copper. In making a 50 per cent. matte out of a 3.5 per cent. ore, with the best possible smelting, and with the furnace in the cleanest and most regular condition, the slag, though without appreciable admixture of matte, will still in itself contain at least 0.7 per cent. of copper. Any deviation from the best possible smelting mixture, any chilling or scaffolding in the furnace, caused by a leaking water-jacket or neglect of duty by the furnace-men, or any of the many accidents always happening at a furnace, will result in a slag still richer in copper. With a normal slag containing *not less* than 0.7 per cent. of copper, the utmost vigilance will be required to keep the average copper contents of the slag produced down to 0.8 per cent., or to only one-seventh higher than in the purest slag obtainable by this method of smelting.

By this method of smelting, under the conditions named, viz., a roasted ore, containing, on an average, say, 3.5 per cent. copper; a relatively small amount of 50 per cent. matte produced; the average addition of only, say, 10 per cent. of *barren* quartz-rock as flux, and about 7 per cent. of ash from the coke entering the slag, I feel sure that of the 3.5 per cent. of copper contained in the ore, only about 2.7 per cent. will be found in the matte produced. In other words, besides the loss of copper in the flue-dust, on account of the abnormal quantity of dust produced by the high blast-pressure, and amounting to

2.5 to 3 per cent. of the total copper in the charge, this quick concentration of copper into a high-grade matte results in the further loss of about 23 per cent. of the copper contained in the roasted ore brought to the smelting-works.

The simple and rapid method of smelting employed by this company whereby a low-grade ore is at once transformed into a marketable high-grade product, appears at first sight eminently practical and successful. But a method, however simple and convenient, which, for the sake of such simplicity and convenience, sacrifices, in the smelting alone, 25 per cent. of the metal forming the object of the industry, and necessitates a further serious loss at the mine by discarding from the material already mined, hoisted and crushed, much of the richest ore, simply because it is slaty, and the slate would be troublesome in the smelting-process (as the same is practiced, at least, in the Herreshoff furnace at the Isabella works), such a method appears to me too wasteful for the most economical treatment of large amounts of the low-grade ores, which are of necessity the basis of operations at Ducktown.

About 17 per cent. of Middlesborough, Ky., coke are required to smelt the charge of ore and flux. This coke is the cheapest obtainable at Ducktown under present conditions as to railroad freights, and has been proved, by actual trial in the furnace, equal, if not slightly superior, to some Pocahontas coke used by the Ducktown Company, and vastly superior to any Alabama coke obtainable. Good steam-coal can be had at reasonable prices from Middlesborough, Ky., as well as from the Coal Creek mines in Tennessee. Either of these coals has little ash, makes few clinkers, and is an excellent steam-coal. The price of coal varies, according to grade, from $1.50 to $2.05 per ton of 2000 pounds, delivered on the railroad cars at any one of the local stations of the Marietta and North Georgia railroad. The price of the Middlesborough coke, delivered in the same way, is $3.05 per ton. These prices are likely to become, through competition, lower, rather than higher.

Wood (hard wood as well as pine and fir) can be delivered at the works in any desired quantity for roasting at about $1.50 per cord. The quartz-flux has been furnished to the Ducktown Company thus far by the owners of teams, who collect the white quartz boulders scattered over the surface of the lands

owned by the company, and deliver the material so obtained on the cars of the company at some point on its railroad. The cost of the flux at the works has been hitherto about 60 cents per ton. Already, however, this float-quartz is getting scarce in the immediate neighborhood of the works, and the price will necessarily increase somewhat in the future. The collection of twenty to thirty tons per day of float-quartz will quickly clear the surface of a large territory.

The Ducktown company has at present two furnaces in continuous operation, smelting together about 200 tons of ore every 24 hours, and capable of treating a still larger quantity when driven to maximum capacity. It has also a small circular water-jacketed cupola copper-furnace about 3 feet in diameter and of the well-known "Arizona" type, with inside crucible and provided with drop-bottom—a remnant from the experiments upon a combined leaching- and smelting-process, referred to above in the history of the district. This furnace is occasionally used for the concentration of matte, occasionally produced, which is too low in copper for shipment to the eastern refineries. When so used, the furnace has produced from roasted low-grade matte much very pure black copper and a smaller amount of very high-grade matte containing from 70 to 80 per cent. of copper.

VIII.—Method of Treatment of the Ducktown Ores Recommended by the Writer.

The Ducktown ore-deposits are of such large dimensions, and their continuity in depth is so reasonably certain, that no doubt need be entertained as to the available quantity of the ore when it has been proven that the quality of the ore is not too poor for the profitable mining and subsequent roasting and smelting. It is the quality of the ore, not the quantity, which is the doubtful factor in the working of any particular Ducktown mine.

After the average lowest percentage in copper, which may be relied on for large amounts of ore produced from any particular ore-deposit, has been ascertained, it becomes a matter of calculation how large the daily capacity of the mine and the roasting- and smelting-plants must be and to what maximum

figures of cost the mining, roasting and smelting operations must be reduced per ton of ore (or per day for so many tons of ore) to leave the necessary margin of profit on the invested capital.

It is self-evident that the lower the cost of mining, roasting and smelting per ton of ore the less copper the ore need contain in order to leave this margin of profit.

Other conditions remaining the same, it is also self-evident that, up to a certain limit, the larger the daily output of the mine the less will be the cost of mining per ton of ore.

The cost of roasting per ton is probably least affected by any increase in the plant and daily product.

But the cost of smelting, on the other hand, is very much influenced by the size of the furnace used. A large furnace will effect a decided saving in fuel and labor over a smaller one, besides allowing usually longer campaigns and requiring less expense for repairs and blowing in and out.

A furnace capable of smelting about 200 to 240 tons (2000 lbs.) of roasted Ducktown ore, or of an easily-smelting mixture, appears to me to reach about the limit of practicable enlargement of the elliptical or rectangular water-jacketed type. A furnace of this kind, 120 inches long by 42 inches wide at the tuyere-level, with 16 tuyeres of 4 inches diameter, melted easily in a trial-run at the rate of 250 tons in 24 hours, with a blast-pressure of 7 to 8 ounces in the blast-main (not at the tuyere-holes, where the blast pressure probably did not exceed 5 to 6 ounces, or even less).

Transportation of ore and fuel on a large scale must be done over tramways. In deciding on the gauge of a tramway-system, it will be found most convenient and conducive to economy in many ways to have the same gauge for all tramways of the system, whether in the mine underground or above ground, inside or outside the works, or on the line of railway connecting the mine with the roasting- and smelting-plants, and to make this gauge that of the side-tracks at the main railroad over which supplies of fuel, etc., are received and the finished product is shipped to market.

Considering all this, it becomes clear that a certain minimum daily output and treatment of ore will be required to allow either the treatment, without loss, of a certain lowest

grade of ore, or to obtain the largest margin of profit on each ton of ore mined and treated.

Besides the cost of mining, roasting, smelting and transportation on the one hand, we must take into consideration, on the other hand, in these calculations the price to be obtained for the marketable product, *i.e.*, the price of copper.

All other factors of the calculation remaining the same, the lower the price of copper the larger must be the daily output of ore from the mine and the capacity of the roasting- and smelting-works for allowing either the same aggregate profit in the same time, or to permit the profitable treatment of an equally low-grade of ore.

In the judgment of the writer a mining and smelting enterprise at Ducktown should not be undertaken on any smaller scale than an output sustaining a regular daily treatment of at least 150 tons of roasted ore. As this is considerably less than the weight of the crude ore, and as the mines are worked only six days in the week, while the smelting-works run continuously, it will be necessary to maintain at least a daily output at the mine of 200 tons per day, or of 5000 to 5200 tons of ore per month.

With such an output, while constantly enlarging the advance of the opening- or dead-work of the mine over the work of ore-extraction, the cost of mining in the Ducktown mines may be kept within the limit of 80 cents per ton of ore, mined, raised to the surface, crushed and delivered in the ore-bins, ready to be drawn off, through chutes, into the dump-cars, which are to convey it over the tramway-system to the roasting-plant.

The ore should be crushed in a crusher sufficiently large and powerful so that a mine-carful ($1\frac{1}{4}$ to $1\frac{1}{2}$ tons) of ore may be dumped at once into the hopper feeding the crusher without choking the machinery. The jaws of the crusher should be set from $\frac{3}{4}$ to 2 inches apart ($1\frac{1}{4}$-inch eccentric to the pitman of the crusher). After passing the crusher the ore should be screened over a screen having round holes of about $\frac{3}{8}$-inch diameter. The "fines" passing through this screen should fall into a separate ore-bin, to be roasted separately in fine-ore roasting-furnaces (with the exception of the small amount of "fines" required for the floor and the cover of the coarse ore when roasted in stalls).

The coarse ore (over $\frac{3}{8}$ inch in diameter) should pass from the screen upon a rather narrow picking-table, sufficiently long to allow enough pickers, sitting on both sides, to pick out all the barren slate and all the slaty and quartzy ore.

The barren rock should be kept separate, and, being crushed to the proper size, would make excellent ballast for the road-bed of the company's railway system.

The slaty and quartzy ores, on the other hand, should be dropped into a special ore-bin, above which the picking-table would be located.

This picking-table would be really an endless conveyor, either a broad rubber belt made for the purpose, or one of the overlapping sectional table-conveyors, carried on link-chains and sprocket-wheels, such as are made by the Jeffreys Link Belt Machinery Company and other manufacturers of link-belt machinery. At the end of this conveyor or picking-table the ore from which all the barren rock, and the greater part of the slaty and quartzy ore, had been removed, would drop into a separate ore-bin, as culled basic- or pyrrhotite-ore. This arrangement will give three classes of ore:

1. *Fines*, which will represent, more or less closely, the average of the mined vein-material.

2. *Slaty ore*, which would carry an excess of quartz and slate, or hornblendic rock, and which would be a siliceous ore, and after roasting could be used as a flux.

3. *Pyrrhotite-ore*, which would be about of the same character as the ore now smelted by the Ducktown company, the *slaty ore*, as well as the barren slate rock, being now rejected and wasted by that company.

Besides these three kinds of ore, crushed barren rock would be obtained, which should be utilized for the ballasting of the railway-system, or for concrete masonry, or any other useful purpose, or put back into the mine for the filling of the stopes, if such filling should be necessary.

The slaty ores and the pyrrhotite-ores should be kept separate and roasted separately, as they require different treatment in roasting, and different quantities of wood and air, to obtain the best possible roast on either kind of ore.*

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* The cost of mining, crushing, and culling 200 tons of ore per day is estimated as follows:

For roasting the coarse ore, heap-roasting is to be condemned without hesitation *for the Ducktown district.* No civilized community ought to be afflicted with the nuisance resulting from the continuous dissemination of scores of tons of sulphurous acid gas daily through that part of the atmosphere nearest the surface of the ground. Experience has shown that the only mitigation of the damages caused by the gases produced in the daily roasting of hundreds of tons of sulphuret-ores is the delivery of these gases from high chimneys into the upper strata of the atmosphere, so that they may become sufficiently diluted

Labor.	Per day.	Total.	
1 mine-foreman,	$3.00	$3.00	
2 shift-bosses,	1.75	3.50	
10 air-drill runners,	1.50	15.00	
10 helpers,	1.25	12.50	
12 trammers and loaders,	1.25	15.00	
2 station-men underground,	1.25	2.50	
1 carpenter and lumberman,	1.50	1.50	
1 helper,	1.25	1.25	
½ machinist,	2.50	1.25	
1 machinist-helper,	1.25	1.25	
2 station-men at top,	1.25	2.50	
4 top-trammers,	1.00	4.00	
2 blacksmiths,	1.75	3.50	
2 strikers,	1.15	2.30	
2 engine-drivers,	1.35	2.70	
1 engine-driver,	1.25	1.25	
2 firemen,	1.20	2.40	
1 culling-boss,	1.10	1.10	
8 boys culling,	.50	4.00	
2 men culling,	1.00	2.00	
Total labor,			$82.50
Supplies and Repairs.			
6 tons of coal at $1.70,			10.20
Oil, lamps, lubricants,			5.00
Tools, steel, etc.,			7.50
Castings, rails, cars, etc.,			5.00
Ropes, mine-timber, etc.,			2.50
200 pounds of 50 per cent. dynamite at $0.13,			26.00
Electric fuse,			2.00
Electric blasting-apparatus, etc.,			.80
Renewal of crushing- and culling-machinery,			5.00
Total,			$146.50

This leaves $13.50 per day for contingencies, to bring the cost up to $160 for 200 tons, or $0.80 per ton.

to be relatively harmless to vegetation before reaching the ground.

It is painful to contemplate the waste of about 20 per cent. of sulphur (*i.e.*, of 40 tons of sulphur per day) from each 200 tons of ore roasted. If this sulphur were only present in more concentrated form and capable of economical utilization, its value would greatly exceed that of the copper in the ore; but no practicable method of saving it is now available, and in the present state of our knowledge we can only endeavor to make this unavoidable waste of sulphur as little detrimental as possible to the agriculture and the forests of the surrounding country. To this end, the ore must be roasted by a method which permits the collection of the roast-gases, or at least the largest part of them, in a chimney which will deliver them at a sufficient height into the atmosphere. The simplest operation of this kind is roasting the coarse ore in stalls, connecting with flues which lead to the chimney. A chimney of sufficient height, built on one of the higher hill-tops in the vicinity, would easily serve as the common draft-producer for all the flues, from the roast-stalls and roast-furnaces as well as from the dust-chambers of the smelting-works.

At the time when the writer built a block of 54 of these stalls, each holding about 25 tons of crude ore, for the roasting of the coarse Polk County ore, doubts were expressed as to the practicability of roasting the Ducktown pyrrhotite-ores in stalls. These stalls were 8 feet long and 7½ feet wide, with side-walls and front-walls 7 feet and rear-walls 8 feet high. Between the rear-walls of the two rows, the central flue, 4 feet wide, and arched over, served as the support of the tramway, on which side-dumping cars, holding about 5 tons of ore each (and so constructed that their loads could be dumped slowly and gradually under the perfect control of one man), brought the crude ore from the mine.

In the front- or outside-wall of these stalls, an opening 3½ feet wide was left for the full height of the stall. Cordwood of the usual length of 4 feet would thus reach across and catch on the jambs of this opening and thus support temporarily the crude ore when the stall was filled. A light dry wall of hard-burnt brick, easily and quickly put up and taken down, was put into the opening in front of the wood, and was braced

against the wooden posts supporting the roof of the shed built over the whole structure, so as to prevent the bulging out and upsetting of this temporary flimsy wall by the swelling of the ore when first fired.

Practical results in roasting about 2500 tons of ore in such stalls, with labor totally inexperienced in stall-roasting, leave no doubt of the practicability of roasting the Ducktown ores as well in stalls as in open heaps.

Stall-roasting of pyrrhotite requires closer attention than heap-roasting, especially during the first six to eighteen hours after firing the stall, and in regulating the amount and quality of wood used for firing the roast-stall to the requirements of the kind of ore contained in each stall. But, after a stall has been properly lighted, and carefully attended during the first twelve or twenty-four hours, no further attention need be given to it; and with ore broken in a jaw-crusher with a jaw-opening of $\frac{3}{4}$ to 2 inches wide, a stall of the size indicated above ought to burn out, and be cool enough for drawing, in from ten to fourteen days. For very slaty ore, less than ten days will be sufficient. Such ore will require two to three times as much wood for burning it properly than pyrrhotite-ore; for the latter, great care must be taken to get neither too much wood, nor such as will give too hot a fire; otherwise, serious matting and even slagging of the ore will be the result, instead of roasting. It is simply a question of experience on the part of the men, and especially of the foreman (as it is also in open heap-roasting), whether stall-roasting will be successful or not. Practice may show that the dimensions of the stalls, and some of the details in their construction, may be modified with advantage; but the fear so freely expressed at first at Ducktown, that the ore in these stalls would necessarily all "matte" into a solid lump, and that successful stall-roasting of Ducktown pyrrhotite would prove impossible, has been shown to be absolutely without foundation.

Roasting in stalls will be cheaper, so far as the fuel is concerned. It will take less than $\frac{1}{50}$ of a cord of wood per ton for solid pyrrhotite-ore, and probably less than $\frac{1}{20}$ of a cord for slaty ore. The amount of labor will be about the same as in heap-roasting. While it will take less labor to fill the stall than to build the heap, the emptying of the stall, on account of the small space which makes it less convenient for the men to

work in, will cost a little more than breaking up and removing a burnt heap. The difference in the cost of the wood will probably be counterbalanced by the greater expense of repairs of the stalls, so that, as far as the cost of roasting ore is concerned, both methods are about equally cheap and about equally effective, with one exception, that the time of the roasting is reduced in stalls to about one-quarter to one-fifth of the time occupied in open heaps.

A part only of the ore "fines" can be taken care of either in open-heap or in stall-roasting. A thin layer of "fines" may be put on the floor of the stall, and another thin layer on top of the coarse ore. The bottom layer will, in part, receive a good roast, and will protect the bottom of the stall from the matte, which is always liable to form in small quantity during the roast, and accumulates generally at the bottom of the stall.

The larger part of the "fines" produced in mining the ore, as well as in the subsequent crushing of the lump-ore, has to be roasted in furnaces erected for this special purpose. The writer intended to use for this purpose a modification of the improved type of "Hasenclever" furnace, designed and perfected for the roasting of quicksilver-ores on the Pacific coast, and described, years ago, in the *Transactions* of the Institute.* It has always appeared to me, that this style of furnace combined the largest capacity with the smallest amount of labor, and, when properly constructed and carefully handled, permitted as good a control of the roasting temperature, at all stages of the roast, as any of the numerous designs of automatic roasting-furnaces, while being about the least expensive structure for its large capacity. The temperature, as well as the time of roasting, may in this furnace be regulated at will, and according to the varying requirements of the different kinds of ore to be roasted, without altering the speed of any machinery; no stirring or ore-transporting machinery being employed in this furnace. While perfectly confident of the successful and economical working of this furnace, under competent management, in the roasting of the Ducktown fine ores, I have not been able to test this opinion by actual experience, as the furnace of this type in course of erection at the works

* See the paper of Professor Christy on "Quicksilver-Reduction at New Almaden," *Trans.*, xiii., 547.

of the Pittsburgh and Tennessee Copper Co. had not been completed when work was suspended by that company. Such a furnace, as planned and partially erected by the writer, would not cost at Ducktown above $4500, and would have a capacity of roasting from 30 to 40 tons of fine ore in 24 hours. The cost of roasting a ton of fine ore in such a furnace would not exceed, at Ducktown, 40 cents at the outside, including a very liberal allowance for repairs and renewal.*

The fine ore would be roasted in such a furnace more thoroughly than the coarse ore could be roasted by one firing in stalls or open heaps, and would thereby afterwards contribute to a higher grade of the matte produced in the first smelting in the ore-furnace.

The roasting-plant should be located, if possible, on a higher level and adjoining the ore-bins of the smelting-works, so that the tracks of the tramways of the roast-yard, over which the roasted ore is brought to the smelting-works, may be level with the top of these ore-bins. These tracks should run over the whole length of these ore-bins, so that the ore taken from any

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* Cost of roasting in 7 stalls, at 25 tons each, of lump-ore per day, is estimated as follows:

1 roast-yard foreman,	$2.25
3 men preparing 7 emptied stalls, and laying wood-foundation in each, at 50 cents per stall,	3.50
8 men filling 7 stalls, and building front walls ready for firing, at $1.50 per stall,	10.50
14 men emptying 7 stalls at $2.50 per stall,	17.50
4 cords of wood, at $1.55,	6.20
200 hard-burnt brick, at $5.00 per M.,	1.00
Tools, oil, repairs, clay, etc.,	7.55
	$48.50
Unforeseen expenses,	5.00
Total cost for 175 tons,	$53.50

or, 30.57 cents per ton.

The cost of roasting 30 tons of "fines" in a "Hasenclever" furnace is estimated as follows:

2 men, at $1.25,	$2.50
2 men, at $1.00,	2.00
1½ cords wood, at $1.60,	2.40
Tools and repairs,	3.60
Contingencies,	2.00
	$12.00

Total for 30 tons, or 40 cents per ton, $12.00.

Or, $63.50 for 200 tons, or, 31¾ cents per ton of ore.

one stall could be dumped in approximately horizontal layers over the entire area of the ore-bin, thereby insuring a thorough mixing, and uniformity in the composition of the different classes of ore (pyrrhotite-ore, slaty ore, and ore-fines), which should, of course, be kept in different ore-bins, and mixed in weighed quantities for each separate charge of the furnace.

This was the plan aimed at by the writer in the construction of the smelting- and roasting-plants of the Pittsburgh and Tennessee Copper Co., erected for the treatment of the ores of the Polk County mine, but not completed according to his intention and plan.

By separating the Ducktown ores we make two classes, pyrrhotite, or basic ores, and slaty, or siliceous ores; and the roasted ore-fines constitute a third class which (according to the character of the ores of each particular mine or of that portion of an ore-body, which is for the time being the place of mining operations) may be either basic or siliceous, but which will always be nearer neutral than either of the two other main classes. With these three classes of ore a competent smelter will always be able to produce a good smelting-charge, without resorting to the use of *barren* fluxes. As a rule, the Ducktown ore-bodies will be able to furnish the required quantities of each class of ore, as the smelter may call for them. The more the opening work in the mine is ahead of ore-stoping, and the more abundant are the roasting facilities of the works, the easier it will be for the smelting to be done in the best possible manner without the use of barren fluxes, by mixing the different classes of ore in the proper proportions to produce the desired composition of slag.

On account of the large amount of fine ore necessarily produced, and to the end of making as little flue dust as possible in the smelting of the ore into matte, it is important that the smelting be done with as low a blast-pressure as practicable. A reasonably low blast-pressure is also considered conducive to the lowering of the copper contents of the slag.*

A low blast-pressure requires a larger furnace for the treatment of the same amount of ore. With a 6-ounce blast-pressure (or probably somewhat less) at the tuyeres, the rectangular

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* See Peters's *Modern American Methods of Copper Smelting.*

furnace, designed by the writer, having a horizontal sectional area of 120 by 42 inches at the tuyere-level, and 54 by 126 inches at the charge-doors, about 9 feet above the tuyeres, melted easily at the rate of from 180 to 200 tons in twenty-four hours. The analyses of the slag produced in a trial-smelting of about 1000 tons of poorly roasted ore,* and under very adverse conditions occasioned by the faulty construction of the fore-hearth, showed that in a normal smelting, and with the production of a matte containing about 20 per cent. of copper from an ore containing 3 to 3½ per cent. of copper, a slag practically free from copper can easily be produced in such a furnace. One analysis of the slag when producing 12 per cent. matte from a 2.1 per cent. ore did not show even a trace of copper, the ammoniacal solution being entirely colorless. While a slag absolutely free from copper might not be *maintained* during a smelting-campaign, the results obtained in this trial-smelting justify the statement, that the average of the copper in the slag produced in such a furnace would not exceed 0.2 per cent. if the matte, produced from an ore containing 3 per cent. of copper did not contain more than 21 per cent. or 22 per cent. of copper, and if the blast-pressure at the tuyeres, using Middlesborough coke as fuel, did not exceed 6 ounces per square inch. Of course, this statement can be verified only under competent management of the smelting works, with a proper composition of the charge uniformly maintained, and all the other conditions necessary to the normal running of such a furnace properly looked after. But as these conditions are the first requisite to the success of any smelting works, they may be properly supposed to be present.

Such a slag, containing on an average only 0.2 per cent. copper, or less, can, however, be produced only when the concentration of the copper in the form of matte is not carried too far. With the Ducktown ores, a majority of which will not average much above 3 per cent. copper, the limit of this first smelting of roasted ore into matte, should probably not exceed 20 to 22 per cent. copper in the first matte. If the concentration be carried further, and higher-grade matte is made, the loss of copper in the slag will rapidly increase. It has been

* The outcome of the first roasting in stalls by inexperienced hands.

found impossible to reduce the copper in the slag below 0.7 per cent., when concentrating these Ducktown ores in the Herreshoff furnace into matte carrying 50 per cent. of copper.

The low-grade matte must be further concentrated to make it marketable, as the transportation and refining charges would be too large on 20 per cent. matte. The low-grade matte must, therefore, be crushed fine to allow of thorough roasting and subsequent conversion, in one more smelting, into a 70 to 80 per cent. matte and into black-copper. That this is feasible by one additional smelting in a cupola-furnace has been proved by practical trial at the Isabella works of the Ducktown company.

The low-grade matte, when tapped from the crucible or fore-hearth of the ore-furnace, could be cast in slabs of a width and thickness which would fit into the opening of the jaws of a Blake crusher, from which the crushed matte could be automatically delivered to either Cornish rolls or a multiple jaw-crusher, which could reduce it to the required fineness. In this state it might be automatically sampled and weighed and delivered into the tram-cars, in which it would then be taken over the railway system of the works to the roasting-plant, to be there subjected to roasting in an automatic roasting-furnace, either of the same (Hasenclever) type as the fine-ore roasting-furnace above alluded to, or of any of the many kinds of automatic roasting-furnaces, used successfully for matte roasting.

To bring the roasted *fine* matte into suitable shape for smelting in a blast-furnace, without the production of too much flue-dust and other objections to the smelting of "fines" in such furnaces, it could be mixed in a common pug-mill with the required quantity of clay to form a rather basic, but good smelting-mixture. From 15 to 20 per cent. of clay would be probably required for this purpose; and an admixture of that amount of clay would be amply sufficient for the subsequent formation of brick in a simple brick-machine. These bricks need only be dried sufficiently to be able to stand being piled up like ordinary brick in a kiln, in which they might be burnt in the same manner as ordinary brick. This burning of the matte-brick would not only make them hard and solid enough for handling and smelting in a cupola-furnace, but would also subject the matte to an additional roasting. I would advise, and intended

to make, experiments to see whether the unroasted matte crushed fine and then formed with the addition of sufficient clay into such brick and burnt in circular brick-kilns, such as are used for the burning of fire-brick and pottery, would not become sufficiently roasted in the burning to do away altogether with any preliminary matte-roasting. In my opinion, this method of roasting, and at the same time bricking the finely crushed matte, will be found practicable. This would also, of course, cheapen the process of roasting and preparing the low-grade matte for concentration-smelting.

This concentration-smelting in a cupola of the matte, thus prepared, would be a very simple process, the charge of the furnace consisting merely of the fuel (coke) and the burnt matte-brick, which would contain the proper smelting mixture, so that no weighing and mixing of different component parts of a charge would be necessary.

The resulting products from this smelting-process would be:

a. High-grade matte (70 per cent. copper and over) and black copper, or nearly all black copper and very little 80 per cent. matte, according to the degree of roasting practiced, which would again depend on whether it would be desired to ship the copper as matte to the eastern refineries, or whether it was the object to produce black copper for subsequent refining at the Ducktown works.

b. Slag rather rich in copper, which would *all* be put back into the large ore-furnaces as soon as cool enough to handle. Here, all the copper contained in this slag would be recovered as matte, with the only exception of not over 0.2 per cent. of the weight of the slag, which could be depended on as the maximum copper-loss in the slag from the ore-furnace. The iron-bearing slag from the matte-concentrating furnace would form a welcome addition to the ore-furnace charge, and could even be used as flux for additional slaty ore. It would practically require no additional fuel to smelt this slag addition to the charge of the ore-furnace, as it would decidedly facilitate the smelting of the ore-mixture and would enable the fuel to carry a larger burden.

c. Flue-dust would be made in small quantity only, as the entire charge would be in lump-form; but whatever quantity is made, it should be carefully collected and added to the mix-

ture of clay and pulverized matte in the pug-mill, and be thus incorporated in the resulting matte-brick.

The losses of copper in this method of treating the Ducktown ores would be contained:

1. In the flue-dust from the ore-furnace. As with a low blast-pressure much less flue-dust would result from melting the same amount of ore, and as a low pressure of blast would not be as liable to carry along into the dust-chamber the heavier ore, as a stronger blast, the loss of copper from this source would be small. It would probably not pay to put this flue-dust into bricks and resmelt it, but this would be a matter of calculation. The loss from this source would certainly not exceed 1 per cent. of the copper contained in the ore.

2. In the slag. This loss would not exceed 0.2 per cent. of the weight of the slag, or about 0.16 per cent. of the weight of the ore. With an average contents of $3\frac{1}{2}$ per cent. of copper in the ore, the loss in the slag from ore-smelting would amount to less than 4.6 per cent. of the copper contained in the ore, and this loss would only be slightly increased, as we have seen above, by the matte-concentration. With a $3\frac{1}{2}$ per cent. ore as a basis, the total loss of copper in the slag produced and thrown away in this method of treatment may reach (for a 0.2 per cent. slag) $5\frac{3}{4}$ per cent. of all the copper contained in the ore.

Allowing 1 per cent. of the copper in the ore as the loss in flue-dust, we may confidently expect a little over 93 per cent. of all the copper contained in the ore, in the shape of a higher grade product, as a result of this method of treatment.

As we have seen, by employing the quick method of concentration-smelting in the Herreshoff furnace, practiced by the Ducktown Co. at the Isabella works, we cannot expect to obtain, in the form of a lower-grade matte, more than 75 per cent. of the copper contained in the ore brought to the smelting-works.

IX.—Comparison of Costs and Results of the two Methods of Ore-Treatment.

The fact that much of the Ducktown sulphuret-ores contains only a small percentage of copper makes it important to the future of the district that these ores should be treated by that method which will permit ores of a lower percentage in copper to be treated without commercial loss.

When we compare the method of quick concentration of the ore *in one smelting operation*, employed by the Ducktown Co., with the method here advocated, we find the main differences to be:

1. In the present method it has been found practically necessary *to cull all the slaty ore* from the vein-material mined and to put it on the waste-dump, together with the barren rock. Disregarding this siliceous ore, and treating only the basic pyrrhotite-ore, makes it necessary to obtain, on an average, at least 10 per cent. of pure quartz-rock as a flux for the roasted ore. This flux not only adds to the expense of smelting, by its own cost, but also increases, by its entire weight, the amount of slag produced. Thus, while it adds no copper whatever to the smelting-charge, it causes an additional loss of copper in the slag, by adding to the weight of the slag produced.

In the proposed method the slaty ores are saved and used as a flux for the basic pyrrhotite-ore, so that no barren flux need be added to the smelting-charge. This secures, in the first place, a saving in the cost of mining, since it utilizes the slaty ore, which is mined and then thrown away by the present method. If 200 tons *of ore* are treated daily, the present method may require the mining, for that purpose, of 250 tons *of ore*, from which amount 50 tons may be culled as slaty ore, and, together with the barren rock, thrown away; while by the proposed method it will be necessary only to mine the 200 tons *of ore* required, together with approximately the same amount of *barren* rock in either case.

It is certainly a low estimate, that the difference in the amount and cost of mining for the same amount of ore actually used in smelting and producing marketable copper will be at least 15 per cent. between the two methods.

It is true that, in the writer's opinion, this disadvantage of the present method might be eliminated by using the slaty ore in place of barren quartz, as a flux for the pyrrhotite-ore. But when this suggestion was made to the superintendent of the Ducktown Co., he replied that experiments made in that direction had shown the slaty ores to be such an "abomination" in smelting in the Herreshoff furnaces used that no further tests or deviations in that direction from the established method were desired.

With a production of 200 tons of ore per day, hoisting the

ore on cages through one shaft in mine-cars, holding 2500 to 3000 pounds of vein-material, and employing a similar arrangement for crushing and culling the ore and loading it on cars, as above advocated by the writer, while employing power-drills in the lead-work, and either power-drilling or hand-drilling in the stopes, the cost of the ore loaded on the cars can easily be kept within the maximum of 80 cents per ton of 2000 pounds when using the slaty ores of the mine. If these ores are thrown away as unfit for the method employed in smelting, at least 10 cents per ton ought to be added to this figure. This estimate provides for a liberal amount of dead-work—preparing the mine for a gradually-increased product.

2. The cost of roasting may be assumed as equal for both methods; and with a capacity of the roasting-plant of 200 tons of ore per day, including "fines," it is perfectly feasible to keep the expenses for roasting within a maximum of 35 cents per ton of crude ore. This figure will leave margin enough for all needed repairs and renewal of tools, etc.

3. In smelting 200 tons of ore per day, the present method consumes about 20 tons of quartz. This, at 60 cents per ton, amounts to $12 per day of extra expense by this method. To smelt 200 tons of ore in Herreshoff furnaces, two such furnaces are required, while under the method proposed, the smelting may easily be done in one furnace. It will obviously cost less, for labor expense certainly, to smelt the same amount of ore in one larger, than in two smaller furnaces. We will, however, neglect this point, as I have not the least doubt of the possibility of constructing a Herreshoff furnace large enough to smelt 200 tons, and also of applying to that furnace the cheaper method of removing the slag in large slag-cars.

By the present method of smelting the ore, we have at least 0.75 per cent. of copper in the slag. On account of the high grade and small volume of matte produced, we must assume the weight of slag at about three-quarters of the weight of the ore plus the full weight of the quartz flux, *i.e.*, with 20 tons quartz and 200 tons ore, we have 195 tons of slag, in which at the minimum contents of 0.75 per cent. copper, 2925 pounds of copper will be contained, which represents a loss of 0.73 of the 3.5 per cent. of copper contained in 200 tons of smelted ore.

By the proposed method, the maximum loss of copper in the

slag can easily be kept at 0.2 per cent. of the weight of the slag. This weight, on account of the larger amounts of matte produced, would be only about $\frac{2}{3}$ of the weight of the ore, but on account of remelting the slag from the matte-furnace together with the ore, will amount to about the full weight of the ore. The loss of copper will be therefore 0.2 per cent. of the weight of the ore.

The cost of smelting 200 tons daily in one furnace will not exceed $1 per ton of ore; and this will allow amply for all necessary repairs and renewals. This is based on the prices of fuel given above, and on a slightly increased cost for labor, over the wages paid at Ducktown at the present time, which, in the writer's opinion, are lower than they can be kept with continued activity in the mining and smelting industry of that place. We will assume the same cost per ton, for the present method, although it is in reality not fair to the proposed method in this comparison to do so.*

* The cost of smelting 200 tons of ore in one furnace by the proposed method is estimated as follows:

Labor.	Per day.	Total.
Smelting superintendent,	$5.00	$5.00
2 furnace-men,	1.75	3.50
2 slag-tappers,	1.25	2.50
2 mules and drivers,	1.75	3.50
6 yardmen removing and crushing matte, and helping generally,	1.00	6.00
4 feeders,	1.50	6.00
4 "	1.25	5.00
8 charge-wheelers,	1.25	10.00
2 engine-drivers,	1.25	2.50
2 firemen and oilers,	1.00	2.00
$\frac{1}{2}$ machinist,	2.50	1.25
2 blacksmiths,*	1.50	3.00
2 strikers,*	1.00	2.00
2 stopper-makers, lamp-tenders, etc.,	1.00	2.00
Total labor,		$54.25
Supplies, etc.		
30 tons of coke, at $3.05,		$91.50
5 tons coal, at $1.70,		8.50
Clay and sand,		1.75
Oil, waste, etc.,		5.00
Castings, tools, repairs,		9.00
		$170.00

* Doing also all general blacksmithing repair-work of the establishment, repair of cars, etc.

4. The cost of transportation of the ore from the mine to the roast-plant, and thence to the smelting-plant; of the furnace-product to the railroad for shipment; of the fuel from the railroad to mine and smelter; and of all incidental work done by the locomotive and train crew over the tramway system of the entire plant, inclusive of maintenance of tramway, may be placed safely at 10 cents per ton of ore, so far as it is not already included in the above estimates of cost of mining, roasting, and smelting of the ore.* No difference is here shown in favor of either method. Of course, the real cost of this item will largely depend on the more or less convenient planning and construction of the plant, and the facilities provided for loading and unloading the railroad cars.

5. The cost of concentrating the low-grade matte produced in the first smelting of the proposed method will be as follows: Crushing the matte automatically to the required fineness, with the aid of the same labor-force which removes the matte from the moulds of the ore-furnace, will not exceed 5 cents per ton of matte, chiefly for renewal of crusher-castings, or on 30 tons of matte, $1.50 per day. Roasting the fine matte in automatic "Hasenclever" furnace, 40 cents per ton, or $12 per day. Cost of not over 6 tons of clay, at 60 cents per ton, delivered at works (same price as quartz), $3.60 per 200 tons of ore. Mixing and bricking the matte and clay, and burning about 8000 bricks at $1.00 per 1000, $8.00 per 200 tons of ore. Smelting 36 tons of burnt matte-brick in small cupola, at (less than) 75 cents per ton, $27 per 200 tons of ore. Total cost of matte concentration for 200 tons of ore:

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* Transportation-expenses per day are estimated as follows:

Engineer,	$1.60
Fireman,	1.00
Train-conductor,	1.75
4 brakemen,	4.00
1½ tons coal, at $1.50,	2.25
Oil, waste, etc.,	1.00
Repairs and renewals,	2.25
1 section-boss,	1.10
2 section-men,	1.80
Ties and rails,	3.25
	$20.00

Crushing matte,	$1.50
Roasting matte,	12.00
Clay,	3.60
Mixing, bricking, and burning,	8.00
Smelting,	27.00
Total for 200 tons ore,	$52.10

6. The general expenses, superintendence, and office expenses may be put in either case at 15 cents per ton for a 200 tons production of ore per day.*

Recapitulation.

A. Total cost per ton of ore, based on a daily output of 200 tons, by the present method:

Mining,	$0.90
Roasting,	0.35
Smelting,	1.00
Flux,	0.06
Transportation,	0.10
General expenses,	0.15
Total per ton of ore,	$2.56

Estimating the value of the pound of copper in the 50-per-cent. matte produced at 6¼ cents, at least at the Ducktown works, which is probably as low as we need expect the price ever to go, this cost of $2.56 per ton would equal the value of very nearly 41 pounds; or 2.05 per cent. of copper in a ton of 2000 pounds. If we add to this 2.05 per cent. the 0.73 per cent. of copper lost by this method in the slag, we have as the minimum percentage of copper in an ore, which can be treated *without loss* by this method, 2.78 per cent. of copper.

B. Total cost per ton of ore, with a daily output of 200 tons, by the proposed method:

* General expenses are estimated as follows per day:

General superintendent,	$11.00
Bookkeeper,	3.00
Time-keeper and weigh-master,	2.00
Assistant to general superintendent, doing surveying and laboratory-work,	3.00
Assistant to same,	1.00
Office and laboratory-expenses,	5.00
Incidental expenses and taxes,	5.00
Total,	$30.00

Mining,	$0.80
Roasting,	0.35
Smelting,	1.00
Matte concentration,	0.26
Transportation,	0.10
General expenses,	0.15
Total cost per ton of ore,	$2.66

Estimating the value of a pound of copper, at the Ducktown works, at 6¼ cents, in the shape of black copper, or 75 to 80 per cent. matte (*i.e.*, at only the same value, at which we estimated it for the 50 per cent. matte of the present method), the cost of $2.66 per ton of ore will represent 42½ pounds of copper; or say 2.13 per cent. To this must be added the 0.2 per cent. copper lost in the slag; and we have a 2.33 per cent. ore as the lowest grade, which we will be safe in treating without a loss by this method, against a 2.78 per cent. ore, by the present method.

In the above comparison I have been very careful to favor the present method in every respect. It is probable, indeed, that under the present method, the ores of a Ducktown mine must contain at least 3 per cent. of copper on an average, to insure against loss, even with competent management and an adequate plant of 200 tons' daily capacity.

On the other hand, under the proposed method, in an adequate plant of 200 tons' daily capacity, and with competent management, 2⅓ per cent of copper in the average ore would insure the enterprise against loss, even with the lowest price of copper, which may be anticipated.

This difference of half of one per cent. of copper in the ore is a vital one with Ducktown mines. For many of them it represents the difference between indifferent success, or failure, on the one hand, and complete success on the other.

X.—Remarks.

We have seen that the Ducktown ore-bodies contain immense quantities of sulphuret ores. The main metallic component of these ores being pyrrhotite, they are *without value*, at least at present, and probably for a long time to come, as *sulphur-ores*.

They are, as a rule, low-grade copper ores. Some of the ore-bodies are too low in copper, to be profitably worked for the

production of that metal. But some, probably many, of these mines contain immense ore-bodies sufficiently rich in copper to be made the basis of a large and profitable mining and smelting industry, with properly planned and equipped plants, and under competent management. Mining and smelting *on a small scale* must inevitably lead to failure at the Ducktown mines. The smallest plant, which in my judgment would have any assurance of success, even with the best plant and with competent management, ought to be able to treat not less than 150 to 200 tons of ore per day. Such a plant, as we have seen, would have an assurance of success even with a $2\frac{1}{3}$ per cent. ore.

With a plant capable of treating 400 or 600 tons of ore, instead of 200 tons, the cost of mining would be easily reduced to 50 cents per ton; the cost of roasting to 30 cents; and the cost of smelting to 80 cents; and that of transportation and general expenses to an aggregate of not over 20 cents per ton. Thus with such a plant the total cost need not exceed $1.80 per ton, or, at $6\frac{1}{4}$ cents per pound of copper, to $28\frac{8}{10}$ pounds or to 1.44 per cent. copper, and with the losses in smelting added, to 1.64 per cent. copper in ore, as the limit of treatment without loss.

If the cost of roasting the ore could be avoided by *pyritic smelting*, this method of ore-treatment would seem to be eminently applicable to such ores as those of Ducktown. Of course, the cost of smelting by this method would have to be about the same as that of the smelting ordinarily done in the largest furnaces. In any event, pyritic smelting would have to be less expensive than the combined roasting and smelting process, as practiced at the present time; and the slag would have to be of the same uniformly clean nature, free from copper, as it is possible to make it by ordinary smelting-methods. To do this, the *regular* running of the furnace in pyritic smelting will have to be under the same easy and complete control of the experienced smelter. Judging from the nature of the process, I am inclined to be skeptical as to its fulfilling this condition; but I would strongly recommend experiments with pyritic smelting on the Ducktown ores, when such are practicable.

The future of the district as a contributor to the copper production of the United States seems assured. One smelting-

works treating daily about 200 tons of ore, is turning out a steady supply of 50 per cent. matte, which is at present shipped east and refined, and which has secured a deserved reputation for the purity of the Ducktown copper, by its freedom from arsenic and antimony.

A second smelting-plant of about the same capacity (200 tons of ore) is nearly completed; and the additional expenditure of a small amount of capital would make it one of the most conveniently constructed plants of its kind, capable of producing copper from a $2\frac{1}{3}$ per cent. copper-ore with assured profit.

The combined product of the two plants, producing about 12 to 14 tons of metallic copper together per day, ought to be sufficient to justify the refining of the copper at Ducktown, which would doubtless be more profitable than shipping the unfinished product to refining-works elsewhere.

The Ducktown copper-mines appear certainly to have an assured and prosperous future before them, especially if the ores are treated in the most rational and profitable, instead of the easiest and most convenient way.

The Assay of Silver Sulphides.

BY H. VAN F. FURMAN, DENVER, COLO.

(Atlanta Meeting, October, 1895.)

There has been considerable discussion of late as to the best method of determining the silver-contents of sulphides of silver resulting from the leaching of silver-ores, and also as to the relative merits of the crucible- and the scorification-method for the determination of the silver-contents of ores.

Owing to the great depression in the silver-market, these are questions of considerable importance, at the present time, to both the producer and the smelter and refiner.

In the hope of throwing some further light on this subject, the following experiments were recently made by the writer:

A few ounces of silver sulphide were prepared by dissolving quite pure silver chloride in sodium hyposulphite and precipitating the silver as a sulphide by the addition of sodium sulphide to the solution. After filtration and slight washing, the precipitate was dried, ground, and thoroughly mixed; and the percentage of silver was then determined by carefully weighing out several portions of 0.05 A. T.* each, on the assay-balance used for weighing the silver buttons resulting from assays; dissolving each portion in nitric acid of 27° Beaumé; boiling to expel the red fumes; diluting with distilled water to 200 c.c., and titrating with a standard solution of potassium sulphocyanate; adding about 1 c.c. of a strong solution of ammonium ferric alum as an indicator (Volhard's method). This method was adopted because the Gay-Lussac titration with standard salt solution could not be used on account of the sulphur set free during solution and the consequent cloudiness of the solution. This free sulphur would also have interfered with the gravimetric determination (precipitating the silver as a chloride and weighing it as such). Volhard's method, provided copper is absent, gives very correct results, and has been used by the writer for years with entire satisfaction. The average silver-contents of the sulphides, as thus shown by several closely agreeing determinations, was 19,693 ounces per ton of 2000 pounds.†

The silver was next determined by the combination method as follows: Two portions of 0.05 A. T. each were dissolved in nitric acid (27° B.), and after boiling out the red fumes each solution was diluted to 300 c.c. with distilled water, and 110 c.c. of normal salt solution (1 c.c. = 10 milligrammes silver) were added, and the solution was vigorously stirred. After allowing the precipitate to settle slightly, 10 c.c. of a strong solution of lead acetate were added, and then 1 c.c. of strong sulphuric acid, the latter drop by drop. After stirring, the solution was allowed to settle over night. The precipitate was

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* SECRETARY'S NOTE.—The assay-ton (A. T.) contains 29,166.66 milligrammes, which is the number of Troy ounces in a ton of 2000 pounds avoirdupois.

† The results have been reduced to ounces per ton, rather than percentages, for the sake of uniformity and convenience. One ounce per ton is 0.00343 per cent.

filtered off through a heavy filter-paper and dried. The precipitate was enveloped in the filter, placed in a 2.5-inch scorifier, and the filter burned by placing it in front of the muffle-furnace. After the filter-paper had been consumed, the scorifier was placed back a few inches into the muffle, to consume all the carbon. After cooling, the residue in the scorifier was mixed with 8 grammes of litharge; 20 grammes of test-lead and 0.5 gramme of borax-glass were added as a cover, and the charge was carefully scorified. The resulting lead buttons (weighing about 10 grammes each) were cupelled in front of the muffle, and the resulting silver buttons were weighed, the result being: *a*, 19,590 ounces per ton; *b*, 19,625 ounces per ton; average, 19,608; average per cent. of the silver present thus found, 99.56.

The *a* assay was made on the pure material, while to the *b* assay were added (in order to introduce such elements as are liable to be present in the sulphides as produced commercially) copper, 0.5; arsenic acid, 0.5; and antimony oxide, 0.5 gramme.

The following table gives the results of the different determinations by both scorification- and crucible-assay:

Comparison of Scorification- and Crucible-Assays.

Scorification Method.				Crucible Method.			
No.	Amount Sulphides taken.	Result. Ounces per ton.	Result. Per cent. of Ag present.	No.	Amount Sulphides taken.	Result. Ounces per ton.	Result. Percent. of Ag present.
					Grammes.		
1	0.05 A. T.	19,576	99.41	1	0.5	19,556	99.20
2	0.05 "	19,714	100.11	2	0.5	19,446	98.74
3	0.05 "	19,562	99.33	3	0.5	19,501	99.02
4	0.05 "	19,573	99.40	4	0.25	19,200	97.49
5	0.05 "	19,675	99.91	5	0.5	18,734	95.13
6	0.5 grammes.	19,547	99.25	6	0.5	19,396	98.49
7	0.5 "	19,406	98.54	7	0.5	19,416	98.59
8	0.5 "	19,611	99.58	8	0.5	19,506	99.05
Average		19,583	99.44	Average		19,344	98.22

Remarks on the Scorification-Assays.—The assays were run in the usual manner, from 30 to 40 grammes of test-lead and from

0.5 to 1 gramme of borax-glass being used. In addition to the regular fluxes, there were added the following reagents:

No. 1. Grammes: 0.5 of Sb_2O_4, 0.5 of As_2O_5, and 0.5 of Cu. The button, after cupellation, weighed 19,764 milligrammes; but, as it showed the presence of copper, it was recupelled with 1 gramme of lead, and the weight of this second button is the result reported in the table.

No. 2. The charge was the same as in No. 1. The button weighed 19,841 milligrammes, and was dissolved in nitric acid, the solution showing the presence of copper. The result, as reported in the table, was obtained by titration of the solution from the button with standard sulphocyanate solution, which method would necessarily give a high result, owing to the presence of copper.

No. 3. This was run without the addition of fluxes other than the regular charge of test-lead and borax.

No. 4. Same as No. 3.

No. 5. Added, grammes: $BaSO_4$, 0.5; Sb_2O_4, 0.5; FeS, 0.4; Cu, 0.5; Zn, 0.5.

No. 6. Added 0.5 gramme of Cu.

No. 7. Added 0.5 gramme of Zn.

No. 8. Added 0.5 gramme of FeS.

Remarks on the Crucible-Assays.—The fusions were run in 10-gramme Denver crucibles, the fusion being performed in the muffle-furnace. The regular charges of litharge, sodium bicarbonate, borax-glass, lead flux and nails were added. The time of fusion was about thirty minutes in each case. In addition to the regular fluxes there were added the following reagents:

No. 1. Added 0.5 A. T. of pure SiO_2.

No. 2. Added, grammes: SiO_2, 5; Sb_2O_4, 1; As_2O_5, 1; Cu, 1; S, 1.

No. 3. Added 0.5 A. T. of pure SiO_2.

No. 4. The same as No. 3.

No. 5. Added grammes: SiO_2, 5; $BaSO_4$, 5; Fe_2O_3, 5.

No. 6. Added 14 grammes of pure SiO_2.

No. 7. Added, grammes: SiO_2, 5; Zn, 1; S, 1; Cu, 0.5.

No. 8. Added, grammes: SiO_2, 5; $BaSO_4$, 5; Fe_2O_3, 5.

In all the assays the same precautions were adopted as would ordinarily be observed in careful commercial work.

The crucible-assays were made, not because this was con-

sidered the proper method for the assay of high-grade silver sulphides, but in order to obtain comparative results by this and the scorification-method.

Conclusions.

It may be objected that the above determinations are too few in number to base conclusions upon. However, it is proper to say that these results but tend to confirm the previous experience of the writer.

The combination-method is by far the most accurate, and yields the most concordant results. On material containing much copper (such as the sulphides produced in the Russell process) this method yields by far the best results, and in the opinion of the writer, is preferable to the corrected scorification-method (where the slag- and cupel-absorption is determined) on account of both the greater accuracy of the results and the difference in time and labor involved. The latter is generally an important consideration in a metallurgical laboratory, where frequently a great number of assays are run in the course of a day.

The regular scorification-assay apparently yields results which may be considered as within the limits of accuracy of a commercial method.

The crucible-method is not suited for the assay of this material.

Concerning the comparative accuracy of the crucible- and scorification-methods, as the writer has observed elsewhere,* the scorification-assay almost invariably yields higher and more uniform results than the crucible-method, especially on sulphides and base ores. The above results appear to confirm this opinion, which was formed after a large series of assays by the regular scorification-method, one crucible-assay being run with each set of scorifications as a check. On some classes of material, such as the oxidized lead-ores and pure siliceous silver-ores, the crucible-method is preferable; but on the great bulk of ores now mined in Colorado and other States the scorification-method is better.

* Discussion of the paper of C. A. Stetefeldt on "The Inaccuracy of the Commercial Assay for Silver," *Trans.*, xxiv., 871. Also, Furman's *Manual of Practical Assaying*, page 123.

Assays of Copper and Copper Matte.

(Made in accordance with the plan suggested by Dr. Albert R. Ledoux, in his paper on "A Uniform Method for the Assay of Copper Materials for Gold and Silver," read at the Bridgeport Meeting, October, 1894, and published, *Trans.*, xxiv., 575.)

(Florida Meeting, March, 1895.)

I.—Introductory Note by the Secretary.

In response to Dr. Ledoux's paper, a large number of metallurgical establishments and individual assayers expressed their willingness to co-operate in the plan he proposed. The necessary samples were prepared as follows:

Matte.—A car-load of matte was put through a Blake crusher, then automatically subdivided into tenths by a Taylor and Brunton sampler, and one-tenth passed through Cornish rolls and through a 12-mesh screen. This was again subdivided by an automatic sampler into tenths, and a final tenth, representing 1 per cent. of the original lot, was finished by hand-sampling on an iron floor until 100 pounds remained. This was pulverized to pass a 40-mesh screen and then thoroughly mixed and divided in the presence of my assistant for distribution to the parties co-operating.

Copper Borings.—These were taken from a lot of anodes, the dip-samples from several batches having been united, remelted and cast into a plate, which was proved by assay to be of uniform quality in its different parts. Borings from this plate were intimately mixed and divided in the presence of my assistant.

Samples were sent out November 1, 1894, with the following letter:

New York City, November 1, 1894.

Samples of matte and copper (about 1 pound each) carefully prepared in the presence of my assistant and sealed with his seal as below, are sent you by express for assay and report according to the plan of Dr. Ledoux.

These samples are constituted about as follows:

	Matte.		Copper.	
	From.	To.	From.	To.
Copper, per cent., ...	50	60	......	......
Gold, oz. per ton, ...	2	3	0.20	0.50
Silver, oz. per ton, ...	100	150	140	180

Please assay them and report results to me, together with a full and minute description of the method employed. In the subsequent compilation and publication of results, the names of works or individual assayers will not be published.

Yours truly,

R. W. RAYMOND,
Secretary.

A request was subsequently made that determinations of metallic copper should be reported, also, and this was done in a number of cases, but not in all. The returns from 19 parties, being all that were received up to February 19, 1895, are given in substance below. For convenience of comparison these returns have been tabulated by the Secretary under general headings, and in this tabulation averages have been given of the reported results of each general class where no difference in manipulation or method was stated in the assayer's description. Where two or more determinations have been made with different precautions or details of operation, it is evident that nothing can be gained for science by averaging the results. In such cases, therefore, all the results of each assayer have been given, unless he himself reported also an average as in his judgment the most probable approximation to the truth. In any event, reference should be made to the more detailed reports for the correction of any errors of judgment on the part of the Secretary in this preliminary classification and tabulation.

With regard to these detailed reports, it should be observed here that they have all been more or less condensed by the Secretary. An attempt to carry this condensation still further so as to omit repetitions of the description of apparently identical methods was abandoned, because it seemed better, on the whole, to include all details of manipulation as reported; since possibly small differences in such details might prove to be important in the critical discussion of results. It should be added, however, that the omission in a given report of the mention of a particular detail or precaution should not be taken as a proof that it was not observed. The reports received have varied greatly in respect of fulness of detail, and unquestionably many small particulars carefully given by some operators have been practiced with equal care by others, but not specified in their description, being regarded as matters of course. The descriptions, therefore, must be taken as exhibiting not absolutely

all that each operator did or omitted, but all that he deemed it important to report.

For the information of foreign members, it may be explained that American assayers report silver and gold in ounces Troy per ton of 2000 pounds avoirdupois. One such ton contains 29,166.66 ounces Troy. The assay-ton (A. T.), a weight introduced by Prof. C. F. Chandler, contains 29,166.66 milligrammes; hence each milligramme of silver or gold, obtained by assay from 1 A. T. of sample, represents 1 ounce Troy per ton of 2000 pounds avoirdupois. The ounce Troy is equivalent to 31,103.49 milligrammes, and one ounce per ton, as above, is equivalent to 34.286 grammes per metric ton of 1000 kilogrammes, or 0.00343 per cent.

II.—Assays by Direct Scorification.

Matte.

	Silver. Ounces per ton.	Gold. Ounces per ton.
A,	127.	2.22
B,	135.38	2.35
C,	129.99	2.33
D,	126.89	2.41
E,	128.95	2.09
F,	127.60	......
G,	128.75	2.22
K,	122.88	2.27
L,	131.22	2.28
N,	126.8	2.29
O,	127.44	2.26
Q,	127.02	2.36
S,	126.4	2.27
T,	130.42	2.33

Copper Borings.

	Silver. Ounces per ton.	Gold. Ounce per ton.
B,	159.33	0.42
C,	159.68	0.32
D,	160.55	0.35
E,	159.12	0.40
G,	147.40	......
K,	155.75	0.35
L,	164.05	0.35
N,	154.40	0.30
Q,	156.90	0.40
T,	160.63	0.37

III.—Assays by Crucible.

Matte.

	Silver. Ounces per ton.	Gold. Ounces per ton.
N,	123.6	2.26

IV.—Assays by Combined Wet and Scorification Methods.

Matte.

	Silver. Ounces per ton.	Gold. Ounces per ton.
D,	130.68	2.31
F,	......	2.40
H,	127.60	1.85
I,	125.20	2.26
J,	129.72	2.325
K,	123.03	2.215
Oa,	125.31	2.24
Ob,	128.06	2.26
P,	128.27	2.05
R,	125.95	2.16
S,	126.4	2.27

Copper Borings.

	Silver, Ounces per ton.	Gold. Ounce per ton.
A,	155.34	0.29
D,	160.78	0.24
F,	156.31	0.22
G,	148.50	0.205
H,	157.3	0.21
I,	156.92	0.28
J,	157.04	0.242
K,	153.65	0.317
L,	161.40	0.30
N,	156.1	0.25
Oa,	156.97	0.22
Ob,	159.67	0.22
P,	156.72	0.501
Q,	159.27	0.40
R,	148.78	0.26
S,	159.0	0.28

V.—Assays by Combined Wet and Crucible Process.

Matte.

	Silver. Ounces per ton.	Gold. Ounces per ton.
M,	126.20	2.09

Copper Borings.

	Silver. Ounces per ton.	Gold. Ounce per ton,
M,	161.35	0.42

VI.—Assays for Copper.

Matte.

	Electrolysis. Per cent.	Cyanide. Per cent.	Iodide. Per cent.
A,	55.08	......	......
C,	......	54.8	55.0
D,	55.17	......	......
E,	54.96	......	......
F,	55.02	......	......
G,	54.50	......	......
K,	......	53.7	......
L,	......	52.72	......
N,	54.73	......	......
P,	......	50.55	......
Q,	54.60	54.37	......
R,	......	......	54.52*
S,	55.08	......	......
T	......	50.75	......

Copper Borings.

	Electrolysis. Per cent.	Cyanide. Per cent.	Method not stated. Per cent.
A,	97.45	......	......
D,	97.04	......	......
E,	98.19	......	......
F,	98.45	......	......
N,	97.50	......	......
R,	......	......	97.52
S,	97.37	......	......
T,	......	97.98	......

VII.—Detailed Reports.

A.

Matte.—Scorify 0.1 A. T. with 45 grammes test-lead, getting button of 18 to 20 grammes. Cupel, weigh, part and weigh Au. The difference is Ag. *Ten checks.* For gold assay, the same, except that only one drop of NaCl solution is added, which gives a small button containing enough Ag to part from Au. *Two checks.* Results: Ag, 127 ounces; Au, 2.22 ounces. Copper by electrolysis, 55.08 per cent.

Copper Borings.—Dissolve 0.5 A. T. in HNO_3, precipitate Ag with saturated solution of NaCl, add as a collector saturated solution of PbA and a few drops H_2SO_4; filter, scorify, cupel, weigh, part and weigh Au. The difference is Ag. *Two checks.*

* Method not stated.

Results: Ag, 155.34 ounces; Au, 0.29 ounce ($6.00). Copper by electrolysis, 97.45 per cent.

B.

Four assays of each sample, 10 scorifiers for each assay. Place in each 15 grammes of test-lead free from Ag and Au; add 0.1 A. T. of sample; mix carefully; pour evenly over the mixture 30 grammes test-lead. To the scorifiers containing matte add one gramme borax-glass; no borax added to copper borings. After thorough melting in hot muffle, scorify at low temperature until lead is entirely covered with slag. Then raise heat to make slag perfectly liquid and pour into moulds. After cooling, free lead buttons from slag by hammering and thoroughly melt in hot cupels, after which cupel at low temperature until about half done, then add 10 grammes pure lead and raise heat slightly to the end. Remove buttons; brush; weigh in sets of 10 and part in sets of 10 for Au. Grind in pairs the cupels of each set to pass through 80-mesh sieve. Fuse each pair in crucible with 60 grammes litharge, free from Au and Ag; 40 soda-ash; 30 borax-glass; 3 argol. Cupel the resulting lead buttons, weigh and part in sets of 5 and add results to those of first cupellations.

The assays here reported were commenced and finished in one day without interfering in any way with the regular laboratory work of a large concern. Results in ounces per ton:

	I.		II.		III.		IV.	
	Ag.	Au.	Ag.	Au.	Ag.	Au.	Ag.	Au.
Matte, 1st Cupellation	123.95	2.25	122.75	2.25	124.95	2.25	123.40	2.25
" 2d "	11.50	0.10	12.60	0.10	10.40	0.10	12.00	0.10
Matte, Total Cupellation	135.45	2.35	135.35	2.35	135.35	2.35	135.40	2.35
Borings, 1st Cupellation	144.70	0.34	148.25	0.34	147.42	0.34	149.12	0.34
" 2d "	15.00	0.08	11.51	0.08	11.31	0.08	10.02	0.08
Borings, Total Cupellation	159.70	0.42	159.76	0.42	158.73	0.42	159.14	0.42

B regrets that samples were not taken that assayed from 500 to 600 ounces Ag and 10 to 12 ounces Au, so as to show more plainly the differences in results obtained by various methods.

C.

Matte.—Sample reground to pass a 100-mesh sieve and thoroughly mixed. Three sets of assays, 3 each, as follows: Mix in each scorifier 0.1 A. T. of matte with 20 grammes test-lead and 1 silica; cover with 20 more lead and 0.5 borax-glass. Scorify at medium heat till covered, then raise heat.

First set then cupelled at low heat, feathering down and raising the heat for several minutes at the very last. Scorifier-slag saved, run down in crucible and cupelled. Silver buttons weighed separately and finally together, taking the averages of the three. The three buttons cupelled together with 2 grammes of lead and a proof-silver of about same size was run at same time, losses noted and corrections made for copper left in silver buttons. Buttons parted for gold, using two acids; washed twice, annealed and weighed. Cupels ground up, run down in crucible, cupelled and weighed.

Second and third sets of 3 each scorified respectively two and three times, scorifier-slag saved and assayed and other procedure, as above.

Fourth set of 3, run to test effect of a very cold scorification; heat raised after the test was covered with slag. Two scorifications; other details as above.

Results.

(Ounces per ton.)

No. of Set.	Average of 3 Ag Buttons.	Ag in Scorifier-Slag.	Ag absorbed by Cupellation.	Cu in Ag Buttons by Proof-Assay.	Total Au and Ag.	Au.	Ag.
1	131.40	0.3	3.7	3.03	132.37	2.33	130.04
2	131.60	0.6	3.53	2.95	132.78	2.34	130.44
3	130.00	1.0	3.20	2.23	131.97	2.33	129.64
4	130.66	0.8	4.00	3.30	132.16	2.33	129.83

Copper, by cyanide method, 54.8 per cent.
" " iodide " 55.0 "

Copper Borings.—Two sets of assays, 5 each, using 0.05 A. T. borings with same amounts of test-lead, borax-glass and silica, as above stated, for matte. Both sets scorified twice. Scorifier-slags and cupels assayed, copper and lead in the silver buttons determined by proof-assay, and gold parted as above. The buttons varied from 3 to 6 ounces. The averages are given.

Results.

(Ounces per ton.)

No. of Set.	Average of 5 Ag Buttons.	Ag in Scorifier-Slag.	Ag absorbed in Cupellation.	Cu in Ag Buttons.	Total Au and Ag.	Au.	Ag.
1	156.70	0.8	4.8	2.3	160.	0.32	159.68
2	152.5	0.8	8.7	2.	160.	0.32	159.68

An attempt with the special method of Mr. Whitehead was a failure, giving Ag 155, Au 0.22. It was also found impossible to use 0.1 A. T. of the borings in scorification—four operations not getting rid of sufficient copper.

A hot cupellation drove much more silver into the cupels—in one case 17 ounces.

D.

Two methods employed, scorification and "combination."

Scorification Method.—Several 0.1 A. T. portions weighed and each placed in a 3-inch scorifier with 50 grammes test-lead, half the lead being intimately mixed with the sample and the rest used as a cover. On top of this charge 1 gramme each of borax-glass and silica. Scorification at moderate temperature for about 35 minutes; scorifiers poured just as they were about to slag over. Buttons weighing about 16 grammes each and quite hard, containing considerable Cu, were cleaned from slag and put in 2½-inch scorifiers, with enough lead to make total weight of each 40 grammes, and a little borax-glass. Scorified and poured as before, time about 28 minutes. Buttons (about 11 grammes each) still contained some Cu, but were soft enough to be cupelled. This done at lowest possible temperature and a good fringe of feathers obtained in every case. Cupels moved back to hotter part of muffle just before the "blick," to secure proper cleaning of beads, which would otherwise be coated with remaining traces of litharge and slag. Beads weighed separately and together, and parted (together) for Au with nitric acid (specific gravity 1.16), two boilings; water, three boilings; dried, annealed and weighed. Slags from both scorifications weighed, ground to pass 80-mesh screen, carefully mixed, sampled and assayed by crucible and cupellation. The cupels were united in two lots of 2 and one of 3 in the case of the matte, and

five lots of 2 in the case of the borings, care being taken to make them correspond to their respective silver-beads. The whole cupels were ground through an 80-mesh screen and fused with silica, soda, borax-glass and a reducing-agent in large crucibles and resulting lead-buttons cupelled. Silver thus obtained, together with that recovered from slag, was added to the average of the corresponding beads. The slag from the matte assays contained but a trace of Ag and no Au; the cupels an average of about 6 ounces Ag per ton of matte. The slag from the borings contained 0.7 ounce Ag, and the cupels an average of about 8.5 ounces Ag per ton of borings.

Silver beads obtained by above method always contain copper, for which some assayers make no correction, on the ground that the copper compensates for the silver lost by volatilization. As the amount of copper thus retained is variable, depending upon the nature of the material, the temperature of cupellation, etc., and does not vary proportionately with the volatilization of silver, this practice is scarcely reasonable. In fact, the loss by volatilization is not large, cupellation being properly conducted.

Results by this method, for both matte and borings, are given below.

Combination Method.—This is a combined wet and scorification method.

1. For gold, weigh two portions, 1 A. T. each, into large beakers with clock-glass covers; add 100 c.c. distilled water to each, and then 50 c.c. nitric acid (sp. gr. 1.42). After violent chemical action subsides, add 50 c.c. more acid, and warm beakers until everthing soluble is dissolved. Evaporate somewhat by boiling; dilute each to 500 c.c. with distilled water; add 3 c.c. sulphuric acid (sp. gr. 1.84), and then 10 c.c. of a concentrated solution of lead acetate. Stir briskly, and allow to settle overnight. Next morning, heat solutions in a steam-bath, and filter through rather thick paper. Wash beakers thoroughly with hot water, finally wipe perfectly clean with filter-paper, which is added to contents of filters. Filtrates should be perfectly clear and free from suspended lead sulphate. Dry filters at moderate heat, wrap in thin lead-foil (weighing about 8 grammes), and transfer to 3-inch scorifiers, with enough test-lead to make total weight 50 grammes. Add about 0.5 gramme

borax-glass to each scorifier; start scorification at low temperature; after papers are consumed, raise heat and proceed as usual. Time in the muffle, 35 minutes; weight of buttons (soft, and, of course, free from copper), 13 to 14 grammes. Clean from slag, and cupel at as low a temperature as possible.

In the cases of both matte and borings, the beads contained some silver (2 or 3 mg.), precipitated, possibly, by a trace of chlorine in the lead acetate or the filter-paper. The beads from the borings were parted directly for gold with nitric acid (sp. gr. 1.16), as in the scorification assay. The beads from the matte did not contain enough silver to part properly, and were therefore alloyed with twice their weight of pure silver and then parted. In the case of the matte, a considerable quantity of free sulphur was produced by the action of the nitric acid, but this caused no trouble in the subsequent scorification.

2. For silver, treat in large beakers three 0.5 A. T. portions, with half the quantity of nitric acid and water used for the gold-assay. Otherwise, proceed as for gold; but, before adding the lead acetate, add enough standard salt solution to precipitate all the silver present, and a *little* more, but avoid great excess, because chloride of silver is soluble in excess of sodium chloride. After stirring, to agglomerate the silver chloride, add 10 c.c. of lead acetate solution; stir again, and allow to settle overnight. Filter, wash, etc., as in the gold-assay. The filtrates should be perfectly clear and free from silver. Scorify as in the gold-assay. Cupel at as low temperature as possible, getting litharge feathers completely surrounding the silver bead. "Blick" at a moderately high heat. Assay cupel-bottoms and scorifier-slags as in the scorification method.

In these cases the slags contained: gold, none; silver, 0.60 ounce per ton of matte; 1.06 ounces per ton of borings.

Remarks.—Gold and silver are determined by the combination method on separate portions, because of the danger of dissolving gold by mixtures of salt with excess of nitric acid, and also because small amounts of gold alloyed with large amounts of silver are more difficult to part than when the silver is in weight only four or five times the gold.

All the results obtained on both samples have been reported; and as none of them were abnormal, the average has been taken. In the case of the matte, ten scorification-assays were started,

but three of them "ran through," by reason of the corrosive nature of the slag. It is a frequent occurrence to lose *one* out of ten scorifications by this process; but three is an abnormally large number.

Although, as observed, none of the results are abnormal, yet the results of the assays of the borings by scorification vary considerably. To those familiar with work of this class such variations will not seem strange. It is impossible to take an average sample of 0.1 A. T. from a lot of borings, and it is only by averaging a number of such assays that a fair result can be obtained.

In both samples, the determinations of gold by the combination process are about 0.1 ounce per ton below the results of scorification. This means a difference of about $2 per ton in assay value. Recent experiments in this laboratory indicate that when copper containing gold is dissolved in nitric acid, a portion of the gold is also dissolved, but whether by the nitrate of copper solution, or by the nitric acid, or by the nitrous acid formed, has not yet been ascertained. As a consequence, the results by the combination method are always too low, and those of the scorification process are to be preferred.

It is believed, on the other hand, that the combination silver assay is more accurate, for the following reasons:

a. *All* the silver can be precipitated as chloride and filtered out, leaving in the filtrates not a trace which can be detected by evaporating them to dryness, and assaying the residue by scorification.

b. The silver chloride can thus be obtained entirely free from zinc and other substances, which tend to volatilize silver in scorification.

c. Copper is almost wholly removed, and cannot contaminate the final beads.

d. The lead button obtained by scorifying the chloride consists, practically, of nothing but silver, gold, and lead, and can be cupelled at a very low temperature, thus insuring a minimum loss by volatilization of silver.

e. The amount of silver absorbed by the cupel is very much less than in the scorification assay. This is an advantage, even when the cupel buttons are assayed, which is not always the case.

Results, in Ounces per Ton.

MATTE.

Scorification Assay. Silver and Gold. First obtained.	Cupels and Slag absorbed.	Total.	Gold.	Combination Assay. Silver and Gold. First obtained.	Cupels and Slag absorbed.	Total.	Gold.	Electrolysis. Copper. Per cent.
128.30 128.60	5.70	134.15						55.15 55.19
128.00 127.80	6.30	134.20	*a* 2.41	129.24 128.80 129.36	3.86	132.99	*b* 2.31 *b* 2.30	
128.20 128.20 128.40	6.26	134.53						

Scorification Assay:

Average........ 134.30
Less gold...... *c* 2.41
Silver-assay... 131.89

The silver-beads contained copper equivalent to 5 ounces per ton of matte. Deducting this, the corrected silver-assay would be 126.89.

Combination Assay:

132.99
Less gold...... 2.31
Silver-assay... *c* 130.68

The silver-beads by this method contain no copper.

REMARKS.

a. Of this gold, 0.04 was in cupel bottoms and 2.37 in first silver beads.
b. Cupels absorbed 0.005 gold.
c. Best results are by scorification for gold and by combination for silver. Hence the assay-report is: Gold, 2.41; silver, 130.68.

BORINGS.

Scorification Assay. Silver and Gold. First obtained.	Cupels and Slag absorbed.	Total.	Gold.	Combination Assay. Silver and Gold. First obtained.	Cupels and Slag absorbed.	Total.	Gold.	Electrolysis. Copper. Per cent.
156.90 164.00	7.50	157.95		156.20 156.20 155.72	4.98	161.02	*b* 0.24 0.24	97.04 97.05
156.80 150.70	10.80	164.55						
156.90 154.80	8.90	164.75	*a* 0.35					
155.60 150.80	9.30	162.50						
157.80 151.90	8.90	163.75						

Scorification Assay:

Average........ 164.70
Less gold...... *c* 0.35
Silver-assay... 164.35

Silver-beads contained copper equivalent to 3.80 ounces per ton of borings. Deducting this, the corrected silver-assay would be 160.55.

Combination Assay:

161.02
Less gold...... 0.24
Silver-assay... *c* 160.78

No copper in silver-beads.

REMARKS.

a. Cupels and slag contained 0.03, and first beads 0.32 gold.
b. Cupels contained a trace of gold.
c. Best results are by scorification for gold and by combination for silver. Hence the assay-report is: Gold, 0.35; silver, 160.78.

E.

Both samples assayed for gold and silver by scorification, except as to assay of ground cupels, which was made with crucible. Cupellation conducted at lowest possible temperature, but care taken to "blick" the bead at the finish. All chemicals were proved pure. Only a single determination of copper was made on each sample.

Matte.—Mixed in scorifier 0.1 A. T. matte with 1 A. T. granulated lead, and put 1 A. T. lead on top of mixture. No borax. Weight of Au and Ag beads after cupellation, in milligrammes:

12.73	12.70	12.66	12.75	12.62
12.84	12.71	12.58	12.67	lost.

Average 12.695. Gold determined by parting the nine beads, 1.88 milligrammes, or 2.09 in 1 A. T.,* which, deducted from $12.695 \times 10 = 126.95$ total Au and Ag in 1 A. T., leaves 124.86 silver. The cupels were powdered to 80-mesh, mixed, and two assays made of 1 A. T. each, showing 0.23 milligramme Ag per A. T. of cupel material, or 0.409 milligramme Ag absorbed in each assay of 0.1 A. T. of matte. Hence, the assay of the

* SECRETARY'S NOTE.—On examination of the proof containing the reports of other assayers, E. was struck by his own exceptionally low figures for gold in the matte. These had been the result of a single determination, without check, and having still on hand a portion of the original sample, he made three new determinations, following in every particular the method above described, with the following results:

I. Weight of beads in milligrammes: 12.68, 12.72, 12.69, 12.74, 12.68, 12.73, 12.80, 12.76, 12.76, 12.72; average, 12.728. Gold, by parting the ten beads, 2.28.

II. Weight of beads in milligrammes: 12.76, 12.58, 12.74, 12.73, 12.66, 12.74, 12.73, 12.82, 12.62, 12.73; average, 12.711. Gold, by parting the ten beads, 2.30.

III. The beads of the third series were not weighed, but were simply parted as a check on the gold, which weighed 2.28 milligrammes. The average of the three determinations, or 2.29 ounces per ton, consequently represents more fairly than 2.09, as originally reported, the true result from the method; and it has therefore been substituted in the summary of results on page 252. The resulting difference in the figures for silver is practically insignificant.

Concerning the low result originally obtained, E. writes: "I cannot account for it. It is one of the accidents which illustrate the necessity of checks." To point this moral, which seems to me one of the most important that can be drawn from the present inquiry, I have left in the text the erroneous determination here corrected.—R. W. R.

matte shows 124.86 + 4.09 = 128.95 ounces of Ag, and 2.09 ounces Au per ton. Copper, by electrolysis, 54.96 per cent.

Borings.—Scorifiers charged as above. Weight of beads:

15.36	15.39	15.48	15.54	(15.04) F.
15.38	15.44	15.48	15.47	15.41

Average (not including F, which "froze") 15.439. Gold determined by parting all the beads of the upper line together, and all the lower line together, giving in each case 0.2 milligrammes Au. Hence, the borings contain 0.4 Au per ton, which, deducted from weight of beads ($15.439 \times 10 = 154.39$), leaves 153.99 Ag. From numerous experiments under the conditions of this case, I find that when 0.1 A. T. of pure copper is scorified with 2 A. T. of granulated lead, containing 15 milligrammes of Ag, 3.23 per cent. of the Ag is lost. Hence, I conclude that 153.99 is but 96.77 per cent. of the actual contents in the sample. Making this correction, we have:

	Ounces per ton.
Silver, ...	159.12
Gold, ...	0.40

Copper by electrolysis, 98.19 per cent.

F.

Matte. — To determine copper, dried sample at 100° C., weighed out in three 1-gramme portions; dissolved in nitric acid; added 5 c.c. sulphuric acid; evaporated till sulphuric fumes came off strongly; cooled; took up with about 55 c.c. water; added enough weak salt solution to precipitate all the silver; filtered solution through small filter into No. 2 plain beaker, and washed filter out with hot water; made filtrate ammoniacal, and then neutralized with nitric acid, adding 5 c.c. excess of the acid; then diluted solution to about 0.5 inch from top of beaker, and electrolyzed.

To determine silver, weighed sample in 0.1 A. T. portions into scorifiers, scorified once with pure test-lead, and cupelled at as low heat as possible.

To determine gold, ground sample as fine as possible in a coffee-mill, quartered down very carefully, and took 1 A. T. portions. Dissolved in nitric acid; diluted to about 10 fluid ounces; added 5 c.c. lead acetate; boiled until free from red

fumes; filtered and transferred to scorifier; burned off filter-paper; scorified with 1 A. T. test-lead and pure silver, and cupelled; parted the button with dilute nitric acid (one-half acid and one-half water), then with heated concentrated nitric acid, and washed with hot distilled water until washings showed no turbidity with salt solution; gold transferred to an annealing-cup; dried, ignited, and weighed.

Borings.—To determine copper, ground and quartered sample, and weighed out three portions of 1 gramme each, weighed to fourth decimal. Dissolved each separately in nitric acid, in a 32-ounce flask; diluted to about 10 ounces; precipitated silver with salt solution; boiled; cooled to the temperature of the room; diluted to 1 liter, and took one-fortieth (25 c.c.) from each for electrolysis.

To determine silver, dissolved portion, quartered and weighed as above, with nitric acid; diluted to 9 or 10 fluid ounces; added 10 c.c. sulphuric acid; boiled till all red fumes were driven off; precipitated silver with excess of salt solution; added 10 c.c. saturated solution of lead acetate; boiled about five minutes, and allowed to stand over night. In the morning, heated and filtered; transferred filters and precipitates to scorifiers, burning off filter-paper, and scorifying with 1 A. T. test-lead (free from Ag and Au); cupelled, weighed and parted buttons for gold, which is weighed and deducted. The results were then calculated in ounces per ton.

To determine gold, the sample was treated as above described for matte, except that 5 c.c. of sulphuric acid was added with the 5 c.c. of lead acetate.

Results.

	Silver. Ounces per ton.	Gold. Ounces per ton.	Copper. Per cent.
Matte,	127.60	2.40	55.01
	127.60	2.40	55.01
	127.60		55.02
			55.05
Borings,	155.96	0.214	98.43
	156.18	0.219	98.44
	156.32	0.232	98.49
	156.77		

G.

Matte.—To determine copper, dissolved 1 gramme of matte

in 10 c.c. concentrated nitric, and added 7 c.c. concentrated sulphuric acid; evaporated till sulphuric fumes began to come off; diluted with water, filtered, and neutralized with ammonia; added 5 to 10 c.c. excess of nitric acid, and diluted solution to 250 c.c.; then passed the electric current (about $\frac{1}{4}$ ampere) through, using a cone and spiral, until all copper was deposited. The amount of silver in the matte was deducted from the copper thus determined.

To determine silver, scorified 0.1 A. T. with about 50 grammes test-lead, one-half mixed with the matte in the scorifier, the rest used as cover, with 3 to 5 grammes of borax-glass, which may seem excessive, but is found in our experience to permit lowest temperature of operation and give best results. Do not generally clean up the slags unless proportion of Ag is high. With proper work, this slag-loss should be inconsiderable. Cupel in lots of five, placed in a row in the front of the muffle, immediately behind a row of empty cupels, which keep off cooling drafts, and are more important than generally supposed. Temperature kept as low as possible until near the end. Cupels should show plenty of "feather" litharge when finished. We use gas-furnaces; yet, in cupelling mattes and lead bullion, do not rely upon gas-regulation alone to secure the right temperature, but invariably introduce into the back of the muffle, after cupelling is well under way, nests of cold crucibles, the larger exposed surface of which rapidly reduces the temperature to the required point, when the crucibles are removed. The desired heat can thus be secured almost from the start. The silver buttons are weighed separately, and then together.

To determine gold, the buttons are flattened, annealed, and treated in steam-bath, with weak nitric acid (sp. gr. 1.08), until action has ceased. Pour off acid, add strong acid (sp. gr. 1.295), and leave flasks in steam-bath twenty or thirty minutes. Gold will then be found in yellow, perfectly coherent flakes, which can be washed without danger of loss. Consider it important to use weak acid at first and heat gradually in steam-bath, and always anneal buttons after flattening or rolling. Wash gold-flakes with hot water three times.

Not our custom to assay cupels for loss by absorption. With careful work, this loss has been found to average about one per

cent. of the silver contents. Although we are sellers, not buyers, of matte, we never add the cupel-absorption to our first assay results. Thus, if we were selling this matte, we should report it as containing 128.75 ounces Ag per ton, and not 130.68 ounces.

The foregoing is a description of the regular commercial practice in this laboratory.

Borings.—We handle no metallic copper material other than doré bars, and hence have no regular method in use. The results given below were obtained by the method of Prof. Cabell Whitehead.

Results.

Matte: Weight of 14 buttons from 0.1 A. T. each 183.36:

	Ounces per ton.
Average, 13.097, ... =	130.97
Of which, gold, ...	2.22
Silver, ...	128.75
Copper, by electrolysis, per cent., ...	54.50

Four cupels melted, and contents added to buttons, gave an average increase of 1.93 per A. T., according to which the corrected silver-assay would be 130.68 ounces per ton.

Borings.—By Whitehead's method, silver (duplicate test), 148.50; gold (average of two tests, 0.20 and 0.21 respectively), 0.205 ounces per ton. By scorification (average of four unsatisfactory scorifications), silver, 147.4 ounces per ton.

H.

Matte.—Take 1 A. T., moisten to a mud with water; dissolve in strong nitric acid; filter through a porcelain Gooch crucible, without using asbestos-felt. By this means the sulphur which has segregated in numerous small lumps during the digestion with nitric acid, is separated from most of the insoluble residue, which passes through the crucible. A porcelain Gooch crucible is better than a platinum one, as the holes are larger, and permits the residue to run through more easily, while retaining the sulphur. Transfer this sulphur to filter paper, dry and burn in muffle in the same scorifier afterwards used in scorifying residue and chlorides. To the filtrate add about 20 c.c. strong sulphuric acid; evaporate in a dish to a pasty condition, to expel free nitric acid and make residue easy to filter. Take up in water and a little sulphuric acid if necessary; add 5 c.c. of a solution

of lead nitrate (5 c.c. contains 1 gramme of the nitrate) and 5 c.c. of a solution of common salt (5 c.c. contains 0.5 gramme of salt); stir, allow to settle, and filter. The precipitate and residue, which need not be carefully washed, are then dried, the filter-paper burned on a platinum wire, and ash and precipitates transferred to the scorifier, which already contains the residue left after burning the sulphur. Mix and cover with about 70 grammes C. P. granulated lead, and a small piece of borax-glass; scorify, cupel, and part gold and silver in the usual manner.

Borings.—These are treated in a precisely similar manner, except that there is no sulphur to be removed, as in the matte.

Results in Ounces Per Ton.

Sample.	Gold and Silver.	Gold.	Silver
Matte, No. 1, ...	128.45	lost.	......
" " 2, ...	129.85	1.75	128.10
" " 3, ...	129.75	1.90	127.85
" " 4, ...	115.30*	1.46	113.94*
" " 5, ...	123.95*	1.75	122.20*
Copper, No. 1, ...	157.05	0.10	156.95
" " 2, ...	154.70	0.08	154.62
" " 3, ...	157.10	0.10	157.00

Thinking that the addition of a chloride to an acid solution containing nitrates might form *aqua regia*, which would dissolve some gold, I made several assays of both matte and copper, filtering off the insoluble residue carefully before adding the lead nitrate and salt. The results seem to indicate such an action in the case of copper, but not to an equal extent, if at all, in that of matte.

Results in Ounces Per Ton.

Sample.	Gold and Silver.	Gold.	Silver.
Matte, No. 6, ...	124.85	1.90	122.95
" " 7, ...	129.00	1.52	127.48
" " 8, ...	129.15	1.96	127.19
Copper, No. 4, ...	153.65	0.16	153.49
" " 5, ...	158.60	0.21	158.39
" " 6, ...	157.77	0.20	157.57
" " 7, ...	157.90	0.22	157.68

We have had to experiment considerably with these samples, since they are so different (especially the matte) from what we

* Imperfect scorification.

encounter in our regular work that our usual methods were not entirely applicable. We assay black and refined coppers for silver only. They contain no gold, and give no insoluble residues. My ordinary method is as follows:

Dissolve 1 A. T. in least amount of nitric acid; dilute to about 250 c.c. with water; add 5 c.c. of hydrochloric acid; filter, and dry precipitate; burn filter-paper with chloride on platinum wire; drop directly into dry cupel; cover with 6 or 7 grammes lead, and cupel. Results seldom vary over 0.1 ounce for a total contents of 10 to 50 ounces.

I made a couple of assays of this copper sample, as nearly as possible by the above method. Filtered off the insoluble residue, on account of the gold, before adding the hydrochloric acid, and added no lead to increase the amount of the precipitate. Also, on account of the amount of insoluble residue, scorified the residue and chloride, which I would not do in the case of our copper. The results agreed closely with each other, and with the other assays.

Results in Ounces per Ton.

Sample.	Gold and silver.	Gold.	Silver.
Copper, No. 8, ...	156.95	0.20	156.75
Copper, No. 9, ...	156.75	0.20	156.55

In view of all the foregoing, I should report on these samples as follows, taking an average from the results most closely agreeing:

Matte.—Gold 1.85; silver 127.6 ounces per ton.

Borings.—Gold 0.21, silver 157.3 ounces per ton.

I.

Borings.—Dissolved 1 A. T. in a No. 4 Griffin beaker with 140 c.c. commercial nitric acid (sp. gr. 1.345) free from chlorine, added in two portions, 90 and 50 c.c. respectively. After all copper was dissolved, removed cover and washed into beaker; diluted solution with 60 to 70 c.c. water; added 16 c.c. normal salt solution; stirred, and allowed silver chloride to settle overnight. In the morning, filtered through double filters (using Schleicher and Schull's 11 cm. filters, No. 597); carefully cleaned beaker, using a glass rod tipped with rubber; washed the filter three or four times, finally washing the chlo-

ride down into the apex of the funnel; removed and folded filter, enclosing the chloride, placed it in a 2¼-inch Denver scorifier, and burned the paper to ashes, at the front of the muffle. It is better to place the assay in the muffle shortly after starting the fire, and let it remain until the paper has been entirely burned; then take out, and pour over the ashes and chloride about 1 A. T. (measured not weighed) of granulated lead, and scorify in the usual way. Resulting lead-buttons weigh from 3 to 4 grammes. Cupel so as to get litharge "feathers" on the inner side of cupel nearest the door, and when nearly finished set the assay back to the hottest place in the muffle.

Matte.—Dissolve 1 A. T. as described for borings, only adding all the nitric acid at once. Let beaker remain on a warm place or sand-bath, until all sulphur has come up, and has a clear, yellowish appearance; then place it on hottest place until the sulphur melts. Remove beaker, wash off, cover; dilute with 60 to 70 c.c. water; allow to cool, and add the necessary amount of salt solution. (In this case, 13 c.c. Too large an excess should be, of course, avoided; and when the approximate amount of silver is not known, it is best to make a preliminary test.) Proceed as in the case of copper borings.

The parting of gold and silver is done in a No. 1 Royal Berlin porcelain capsule, using 7 c.c. of a mixture of nitric acid and water (1 of acid to 6 or 7 of water), and heating. When all silver is dissolved, pour off solution and add about 4 c.c. of nitric acid (sp. gr. 1.42), and boil; then wash three or four times with distilled water; dry, heat and weigh.

The use of double filters is not necessary always, but the presence of a white precipitate (Sb_2O_3), after dissolving both copper and matte, made it necessary on these samples.

Results, in Ounces per Ton.

	Gold.	Silver.
Matte, ...	2.26	125.20
Borings, ...	0.28	156.92

J.

Borings.—Weigh 1 A. T. into a liter flask; add 150 c.c. nitric acid (22° B.); heat gradually till violent action has ceased and add fresh acid from time to time till solution is complete; boil till red fumes are gone; dilute to 400 c.c.; add 5 c.c. concentrated sulphuric acid and 10 c.c. saturated solution

of lead acetate. When lead sulphate has settled, filter and wash until free from copper. Add to filtrate enough standard solution of sodium bromide to more than precipitate the silver present; warm; stir violently till the silver bromide becomes lumpy and the solution clears up. Filter and wash the silver bromide by decantation until copper is nearly all removed. Reduce the bromide with zinc and hydrochloric acid, dissolving the excess of zinc; wash the silver on the filter free from chloride and bromide; then wash it from the filter by tipping the funnel over a beaker and playing a jet of water upon it. Dissolve in nitric acid (22° B.) and boil until free from nitrous fumes. Filter the solution (containing a trace of undissolved bromide of silver) through the filter from which it was originally washed, and wash the filter free from silver nitrate. To this solution, which should not amount to more than 100 c.c., add 5 c.c. of saturated solution of ferric sulphate and determine silver with a standard solution of sulphocyanide (1 c.c. = 1 mg.). Such a solution is made by dissolving about 0.750 gramme of ammonia sulphocyanide in one liter of water. The solution should be frequently checked with pure silver, though it keeps well when tightly corked. To the solution from which the silver bromide was filtered, add 20 c.c. of lead acetate solution and 10 c.c. of concentrated sulphuric acid; allow the lead sulphate to settle, and when the solution is quite clear, filter, using the paper through which the silver nitrate solution was washed. Wash the precipitate free from copper; reduce with dilute sulphuric acid and zinc; combine with the precipitate containing the gold, and scorify with 0.5 A. T. of lead and 0.5 gramme of borax-glass. Cupel the lead buttons near the front of the muffle, where the cupel will show a small amount of "feather" litharge. The button of silver thus obtained should not represent more than 10 per cent. of the silver present in a 100-ounce bullion. By this method the losses incidental to scorification and cupellation are almost entirely removed, and the "personal equation," so far as it relates to the muffle-temperature, enters only slightly into the assay.

In parting, use two strengths of acid, 22° and 32° B. The buttons should contain at least 6 times as much silver as gold. Flatten them on an anvil, put into a round-bottomed matrass; add 20 c.c. of 22° acid; boil 5 minutes; pour off acid; add

20 c.c. of 32° acid; boil again 5 minutes; thoroughly wash and dry the gold and heat to bright redness in an annealing-cup.

Matte.—The treatment of matte is the same, unless impurities present require, as in this case, especial modification. The method used is as follows:

Dissolved 1 A. T. in 32° nitric acid. No attempt was made to free the sulphur entirely from traces of matte, experiments having proved that in doing so, a large portion of the gold present is taken into solution, either by the nitrous acid formed, or by the nitric and sulphuric acid present at the end of the reaction. Diluted the solution to 400 c.c. and added 5 c.c. sulphuric acid and 10 c.c. lead acetate. When clear, filtered and added enough standard sodium bromide to precipitate all the silver present, also 20 c.c. lead acetate and 10 c.c. sulphuric acid. Allowed to stand till clear; filtered and washed free from copper. Reduced lead sulphate and silver bromide with metallic zinc. Placed in a scorifier, in front of muffle, the filter containing sulphur, lead sulphate and gold, and when paper and sulphur had burned off, added 1 A. T. test-lead and 1 gramme borax and scorified residue. The reduced lead and silver is scorified with the lead button from the gold residue, with the addition of 0.5 A. T. test-lead. The cupellation of this button and the subsequent parting of the gold was the same as in the assay of copper borings, except that the cupels were assayed, and the silver and gold obtained were added to the original weights.

Results in Ounces Per Ton.

Average of four closely agreeing determinations:

	Gold.	Silver.
Borings, ...	0.242	157.04
Matte, ...	2.325	129.72

We regret that Dr. Ledoux did not extend his suggestion so as to include both Russell sulphides and electrolytic slimes—products far more difficult to assay than either matte or bullion.

K.

Results and methods were given in this case in the form of tables and comments as follows:

TABLE I.—*Assays of Matte for Gold and Silver.*

No.	Weight of Ag and Au Button, Mgr.	Ag recovered from Slag, Mgr.	Ag recovered from Cupel, Mgr.	Ag, ounces per ton (corrected).	Ag, ounces per ton. Average.	Au, ounces per ton, Average.
1	122.5	none	1.0	121.24		
2	119.5	none	1.0	118.24		2.26
3	124.5	none	1.0	123.24		
4	124.5	none	1.0	123.24	122.88	
5	125.5	none	1.0	124.20		2.30
6	123.5	none	1.0	122.20		
7	123.5	0.35	2.0	123.60		2.25
8	128.0	0.35	1.0	127.10		
9	....	....	....	122.80		2.20
10	....	....	....	123.30	123.03	2.26
11	....	....	....	123.30		2.20
12	....	....	....	*118.50	....	2.20

Remarks.—Nos. 1 to 4 inclusive were scorification-assays, each on 0.1 A. T. matte, weighed, mixed with 20 grammes test-lead, and covered with same amount of lead and 0.8 gramme borax-glass, and scorified at as low temperature as possible, especially at the start. Each resulting lead button was rescorified with 20 grammes test-lead and 0.2 gramme borax-glass. The second lead-buttons (about 10 grammes each) were cupelled, and, after weighing, the four beads were parted together for gold. The same procedure was followed for Nos. 5 and 6, the two beads of which were parted together. In assays Nos. 7 and 8, the first scorification was made with 50 grammes of lead (half in mixture and half as cover), and the second with 25 grammes of lead, other details being the same as before; and the buttons were rescorified without further fluxing down to about 6 grammes and then cupelled, the two beads being parted together. In every case slags and cupels were saved, the slags from each two sets of assays being ground up together to 40-mesh size and assayed for silver by crucible-assay, and the cupels being ground and assayed separately by the same method.

In Nos. 9 and 10, Whitehead's method was used; in Nos. 11 and 12, the following modification of Whitehead's method: Put in No. 5 beakers two weighed portions, 1 A. T. each; add

* In averaging results No. 12 was omitted, being evidently too low.

to each 100 c.c. water and 50 c.c. nitric acid (sp. gr. 1.42). When action has apparently ceased, add 50 c.c. more acid and expel red fumes by boiling. Cool solutions, and treat separately as follows:

1. *First Portion.*—Add 10 c.c. saturated solution of lead acetate and then 5 c.c. of dilute sulphuric acid; allow precipitate to settle; decant upon a filter, and finally wash precipitate on to the filter; the water used for this purpose will generally be sufficient to remove the copper salts. Dry, filter and precipitate thoroughly in a scorifier. When dry, burn filter in front of muffle, and gently ignite the whole in front part of muffle. Add 20 grammes test-lead and a pinch of borax-glass, and scorify at low temperature. Cupel and part in the usual manner.

2. *Second Portion.*—Add 15 c.c. saturated solution of sodium chloride; stir, and add 20 c.c. saturated solution of lead acetate; allow to settle; then decant, etc., as with the first portion.

TABLE II.—*Assays of Matte for Copper.*

No.	Copper. Per cent.	Copper. Per cent. Average.
1	53.3	
2	53.3	53.2
3	53.5	
4	52.8	
5	54.1	54.2
6	54.3	

Remarks.—Nos. 1 to 4 inclusive were made as follows: Place in a pear-shaped flask with wide neck (capacity 250 c.c.) 0.5 gramme matte, and add 7 c.c. nitric acid (sp. gr. 1.28); boil a few minutes; cool; add 1 c.c. concentrated hydrochloric and 5 c.c. concentrated sulphuric acid; boil until copious sulphuric fumes are evolved; cool; dilute with water to about 50 c.c., and put in four strips of 1 by $\frac{5}{8}$ by $\frac{1}{16}$ inch aluminum foil, free from copper. Place flask on asbestos card-board, heated underneath by a Bunsen burner; boil gently about fifteen minutes, which will generally be enough to insure complete precipitation of the copper; remove from the heat; fill with water; allow to stand a few minutes, and decant clear solution upon a fluted filter. After washing in this way three times, dissolve

in 7 c.c. nitric acid (sp. gr. 1.28) the copper-precipitate in the flask, and any particles which may have passed to the filter; keep solution at boiling-heat until red fumes are gone; then pour off into a similar flask; thoroughly rinse with water the first flask and the aluminum strips, and add washings to the main solution. This should be about 100 c.c. in bulk. Add 20 c.c. strong ammonia, and, after cooling, titrate with standard solution of potassium cyanide. The standard solution is added from a burette, with frequent agitation of the contents of the flask, until one drop just discharges the faint violet tint of the solution. One c.c. of the standard solution is equivalent to 0.005 gramme of copper. To prepare such a solution, dissolve 57 grammes pure potassium cyanide in 1 liter of water, and standardize with C. P. copper foil, proceeding exactly as in the regular analysis, from the point where the copper is precipitated on the aluminum.

In Nos. 5 and 6, essentially the above method was followed, but zinc was substituted for aluminum as a precipitant, strips of commercial zinc to the amount of 7 grammes being used. After copper was completely precipitated, the excess of zinc was dissolved by adding a few c.c. strong sulphuric acid. The use of a filter in decanting the solution is unnecessary, because the copper is not so finely divided as when aluminum is used. But the writer prefers to use aluminum on account of the danger of high results from remains of zinc in the precipitated copper. Moreover, aluminum leaves a clear solution, in which the end-reaction can be plainly seen. This is not always the case with zinc, especially if it contains much lead.

TABLE III.—*Assays of Borings for Gold and Silver.*

No.	Weight of Ag and Au Button, Mg.	Ag recovered from Slag, Mg.	Ag recovered from Cupel, Mg.	Ag, ounces per ton (corrected).	Ag, ounces per ton, Average.	Au, ounces per ton.	Au, ounces per ton, Average.
1	148.0	1.5	5.5	154.65	155.75	0.35	0.35
2	151.5	1.5	2.0	154.65			
3	153.5	0.35	2.0	155.50		0.35	
4	154.7	0.35	3.5	158.20			
5	152.3	....	....	152.0	153.65	0.33	0.317
6	154.2	....	....	153.9		0.30	
7	157.0	....	....	156.7			
8	152.3	....	....	152.0		0.32	

Remarks.—Nos. 1 to 4 inclusive were scorification assays on portions of 0.1 A. T. Three scorifications were run on each portion, 50 grammes test-lead and 0.8 borax-glass being added for the first, 25 lead for the second, and five lead for the third scorification. No. 5 was made by Prof. Cabell Whitehead's method.* In Nos. 6, 7, and 8 the modification of that method described under Table I. was followed, except that in Nos. 6 and 7 the portion of borings taken was 0.5 A. T., and in No. 8 it was 1 A. T.

L.

Matte.—Weigh out ten portions of 0.1 A. T. each, scorify each in a $2\frac{1}{2}$-inch scorifier, mixing with 25 grammes test-lead and 0.5 gramme glass, and covering with 25 lead and 1 fused borax. Start with a bright red heat, continue at a little lower heat until the end; then raise to bright red and pour. Clean off slag, transfer to new scorifiers and rescorify. Cupel each lead button separately, and weigh the resulting ten silver-gold buttons together. Grind up and fuse cupels in five lots of two each, and cupel, adding the five resulting buttons to the ten silver-gold buttons. Charge for fusing two cupels, in grammes: 15 soda, 75 litharge, 5 powdered borax, 2.5 argol, and 12 powdered glass. Mix thoroughly with the ground cupels, and add 15 fused borax as a cover. Part silver-gold buttons in a one-ounce porcelain cup, using first dilute nitric acid (one acid to three water; sp. gr. 1.14). Fill the cup about half full, and heat gently, taking care not to break up the buttons. After ebullition ceases, pour off acid, and treat with concentrated nitric acid (sp. gr. 1.42). Boil

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* Prof. Whitehead's method (see *Jour. of Anal. and Applied Chem.*, vi., 262, and Furman's *Manual of Practical Assaying*, New York, 1893, p. 250), is substantially as follows:

Dissolve 1 to 4 A. T. in a large beaker (500 c.c. capacity) by the gradual addition of strong nitric acid; drive off red fumes by heating in sand-bath; add 50 c.c. saturated solution of lead acetate; stir; add 1 c c. dilute sulphuric acid, and allow lead sulphate to settle. Filter; wash with cold water, dry in scorifier; burn filter-paper; scorify with test-lead; cupel, weigh, and part as usual.

Dilute the filtrate to 1000 c.c.; divide in halves of exactly 500 c.c.; add to each saturated solution of sodium bromide so long as a precipitate forms. A large precipitate of lead bromide collects and envelopes the silver bromide, permitting immediate filtering without loss. Filter; wash with cold water; dry filters and precipitates; brush into small crucibles; mix each with three times its weight of carbonate of soda, and some flour or argol as reducing agent; cover with borax-glass; fuse for lead buttons; cupel and weigh. The two results should agree closely.—R. W. R.

hard until acid becomes colorless; then decant; wash the gold in the cup several times with hot water; dry; transfer to a hot muffle for a few moments; cool and weigh.

The above is out of our usual line of work. We receive no metallic copper, and our matte never carries over 0.08 ounces gold per ton.

Assays of Matte for Gold and Silver.

	Milligrammes.
Weight of ten silver-gold buttons,	130.
" five buttons from cupel absorption,	3.50
" total Au and Ag,	133.50
" gold obtained by parting,	2.28
" silver,	131.22

Hence the matte contained 131.22 ounces Ag and 2.28 ounces Au per ton.

Borings.—These were assayed in the same way, with the following results:

Assays of Borings for Gold and Silver.

	Milligrammes.
Weight of ten silver-gold buttons,	158.4
" five buttons from cupel absorption,	6.0
" total Au and Ag,	164.4
" gold obtained by parting,	0.35
" silver,	164.05

Hence the borings contained 164.05 ounces Ag and 0.35 ounce Au per ton.

I have also tried the borings by the following combination method: Dissolve 1 A. T. in No. 5 Griffin beaker, with 100 c.c. water free from chlorine, and 50 c.c. concentrated nitric acid. After violent action has ceased, add 50 c.c. more of the acid. After copper is all dissolved, heat and boil until free nitric acid has been expelled; dilute with 250 c.c. water free from chlorine; add 5 c.c. concentrated sulphuric acid and 10 c.c. concentrated solution of lead acetate; stir vigorously, and allow to settle over night. Next morning filter, and precipitate silver in filtrate with sodium chloride, and allow this precipitate to settle until next morning. Carefully dry and combine the two precipitates and scorify, cupel, and part, determining also silver and gold absorbed in cupels.

Combination Assay of Borings for Gold and Silver.

	Milligrammes.
Weight of button from scorification and cupellation, . . .	159.4
" " " " cupel-absorption, ...	2.3
" " total Au and Ag, ...	161.7
" " gold obtained by parting, ...	0.3
" " silver, ...	161.4

Determination of Copper.—This was performed on the matte only, as follows: Treated 0.5 gramme matte in a 250 c.c. flask with 10 c.c. nitric and 5 c.c. sulphuric acid; boiled until sulphuric fumes came off; cooled; diluted with 50 c.c. water and 10 c.c. sulphuric acid; added strips of sheet zinc until all copper was precipitated; filled flask with water and decanted several times through filter-paper, which caught any small particles carried over by decantation. Dissolved the copper precipitate in the flask in 8 c.c. concentrated nitric acid and the particles on the filter with 2 c.c. nitric acid, adding this solution to the flask; boiled to expel all free nitric acid; cooled; diluted with water to 150 c.c.; added 10 c.c. ammonia, and when the solution was of the same temperature as the room, titrated with potassium cyanide, taking care to have the same volumes and same time for titration as in standardizing the cyanide solution. The copper solution was filtered while it was still of bluish-purple color, so as to have a clear solution for the end-reaction.

The copper in the matte was thus found to be 54.02 per cent.; and the result reported was 52.72 per cent., 1.3 per cent. being deducted from the figures given by wet assay, to reduce them to the basis of dry assay, according to commercial custom.

M.

Matte was crushed to pass 100-mesh screen. Copper borings were passed over a 10-mesh screen and proportional parts of coarse and fine were taken. To determine silver approximately, I make a preliminary scorification assay on 0.1 or 0.2 A. T., using 2 to 4 A. T. of assay-lead. As soon as it covers casting in mould, I break off slag and scorify again until lead is down to 10 grammes. I stir the copper through the melted lead with an iron rod.

This approximate scorification gave for the matte, 2.345

ounces gold and 113 ounces silver, to which was added 2 ounces loss by table, making 115 ounces silver; and for the borings, gold, less than 1 ounce and silver (including loss by table) 135.2 ounces. These figures are a guide to the amount of salt solution to be added, as below—which I want a little in excess.

For the assay I weigh out 1 A. T. into a beaker 4½ inches deep; add slowly 50 c.c. nitric acid (sp. gr. 1.20), keeping covered with a glass; after energetic action is over, add slowly 50 c.c. nitric acid (sp. gr. 1.4), keeping the beaker on the sand-bath; and then 30 to 50 c.c. (in this case 50 c.c.) more, until action seems to have stopped. Continue evaporation till the copper salts begin to crystallize out and the sulphur is melted, often into a cake; then dilute with boiling water until beaker is two-thirds full, and add a *collector* made of 60 parts Lake Superior red hematite, 30 parts fire-clay and 10 parts of sulphate of iron crystals passed through an 80-mesh sieve. Of this material, kept on hand for the purpose, I use 10 grammes, adding 5 to the solution before filtering and 5 to the filtrate after adding the salt, as below. Some ores dissolve almost entirely, and it is difficult to handle the small residue without loss. I stir it for a few minutes, let it settle a very short time and filter through a 15 cm. close Swedish filter; dry; burn off the sulphur in a cake first and then the paper, using a granite-ware dish that has been pretty thickly coated with iron oxide, and is heated only just hot enough to char the paper. The residue is weighed. (To the filtrate I add 100 mgr. of salt and 5 of collector; filter again and add to the first lot. Test the last filtrate with salt to see that the solution remains clear.)

Weigh into a B crucible with the residue an equal weight of litharge, half-weight of borax-glass, half-weight of flux (composed of carbonate of potash and caustic soda kept in air-tight tin cans), and 0.1 A. T. test-lead, mix together and add on top ½-weight of borax-glass and ¼-weight of flux, and usually an iron nail, from habit. Have tried duplicates with and without the nail, and have sometimes thought I got better results without it. I think if the melt is perfect and hot, a nail is good; but if the work is hurried some of the charge is apt to stick on the upper part of the nail. I usually weigh out only the

lead and litharge, adding the rest by the eye, with a salt cover. If a large amount of iron has remained undissolved, I add a little Florida white sand. My lead-buttons in this case weighed 0.4 A. T. from the borings and 0.5 A. T. for the matte. The slags were perfectly melted, black and glossy.

In cupelling, I heat my cupels twice, that is heat up, cool down, and heat up again, before putting the buttons on; cupel with a few feathers and increase heat at the end, but seldom "sprout" a button.

In parting, I use first, nitric acid of a little less than 1.20 sp. gr. (at our works we made an acid 1.16); then boil again with acid of sp. gr. 1.4; wash out twice with distilled water cold, and then boil with water; wash out and dry, and ignite in annealing-cup or porcelain crucible. If gold is just a speck, I brush out into a capsule and into a small lead cornet, which I run down on a blow-pipe cupel or the furnace. Or, if a glass shows that it looks all right, I weigh the gold direct.

Results in Ounces per Ton.

	Au and Ag in button.	Au and Ag from slags and cupel.	Au in button.	Au from slags, etc.	Silver.
Matte, I., . .	120.50	4.63	1.40	0.09	123.64
" II., . .	123.66	4.63(?)	2.00	0.09(?)	126.20
Borings, I., .	142.10	15.00	0.27	0.07	156.76
" II., .	156.34	5.48	0.37	0.059	161.35

I had trouble with the gas in my furnace in the first results of matte and borings, and the second determination is, in each case, doubtless the best, as far as the weight of button is concerned. I should therefore report the matte as containing, in ounces per ton, silver, 126.20; gold, 2.09; and the borings as as containing, silver, 161.35; gold, 0.42. It will be noticed, however, that I did not determine the second and fourth columns for the second matte-assay, but adopted the figures of the first assay, being pressed for time at the last. In commercial practice, I make no report of cupel- or slag-absorption.

N.

Matte.—The matte being very coarse, all but 1 A. T. of our portion was reground, and put through a 100-mesh sieve.

Assay of Matte for Copper.—Our usual method is: Weigh up

500 mg. on a button-balance; transfer to a No. 1 tall-form beaker; moisten; add 7 c.c. nitric acid (sp. gr. 1.42); set to warm; boil and evaporate to 4 c.c.; dilute to 30 c.c.; boil and filter through 7 cm. No. 0 Munktell's filter into a No. 2 Griffin's beaker; render ammoniacal with only enough ammonia to redissolve copper; acidify with sulphuric acid, and add the equivalent of 4 c.c. concentrated sulphuric acid; dilute to 150 c.c., and electrolyze. We use a dynamo current, with lamps as rheostats, and assay in series. Cylinders are used for precipitation. These are slit open from top to bottom on the side opposite the stem to permit free circulation between the interior and exterior. The spiral is made to touch the bottom of beaker, and the bottom of cylinder is placed $\frac{1}{2}$ inch above it. The assays are set on at night, and in the morning some water is added. The deposition is almost invariably found complete. The copper is bright and clear. The silver is deducted from the copper deposited, an allowance of 0.1 mg. being made for unavoidable traces of chlorine. Duplicates check within 0.3 mg.

With the matte sent us in this case, we did not obtain the usual favorable results. Some loose copper and impurities were deposited. For such material, I do not believe electrolysis to be as good a method for determining copper as some others. By dissolving in nitric acid, boiling down to 6 c.c., diluting and electrolyzing, 54.73 per cent. of copper was obtained. Another method tried was to dissolve in nitric acid and run down to heavy fumes with sulphuric acid; take up with water, precipitate silver with a titrated solution of salt; filter; precipitate copper with zinc; filter, wash, dry, and ignite the copper; dissolve in 4 c.c. nitric acid, and treat further like metallic copper. This method gave 54.73 per cent. likewise. In neither case was the copper as bright and clean as that from our own mattes.

Assay of Borings for Copper.—Weigh up carefully about 500 milligrammes; dissolve in about 5 c.c. water and 4 c.c. nitric acid (sp. gr., 1.42); warm, boil, dilute, and filter out any undissolved residue; render ammoniacal with slight excess; acidify with dilute sulphuric acid, and add the equivalent of 2 c.c. concentrated sulphuric acid in excess; dilute to 150 c.c. and electrolyze. Allow for silver as in matte. Duplicates on uniform

samples check within 0.2 to 0.3 milligramme. The copper in the borings sent was 97.50 per cent.

Assay of Matte for Silver and Gold.—We use the scorification method for silver and the crucible for gold. Formerly we scorified 1 A. T. in 0.1 A. T. lots for gold, but we found the results on low-grade mattes to be the same by both methods.

Put 0.1 A. T. matte in a 2½-inch Denver fire-clay scorifier, with 50 grammes test-lead and a pinch of borax-glass; ⅔ of the lead mixed with the matte, ⅓ for cover. Melt at high heat and scorify hot. Resulting button, 20 to 25 grammes. For second scorification, add 15 grammes test-lead; scorify hot. Resulting button, 12 grammes. Cupel for few "feathers."

For the crucible-assay of matte, make up flux as follows: 120 parts litharge, 25 carbonate of soda, 20 silica, 6 niter. Use 170 grammes flux, 0.25 A. T. matte, and 20 grammes salt cover. Melt in Denver crucible forty-five minutes. Resulting button 18 to 20 grammes. Cupel for few feathers.

Parting for Gold.—Use test-tubes and dilute nitric acid (1 part of sp. gr. 1.14 in 4 parts). Place test-tubes in beaker filled with dilute sulphuric acid and heat. Then boil over alcohol lamp, dilute, and decant; add concentrated nitric acid; boil, dilute, decant; wash twice; transfer to porcelain crucible; dry, ignite, and weigh.

Results in Ounces Per Ton.

Matte.	Ag.	Au.
By scorification, coarse sample,	126.9	2.27
By scorification, reground sample,	126.7	2.30
By crucible,	123.6	2.26

Assay of Borings for Silver and Gold.—To determine silver, we use chloride precipitation. For gold, we formerly used dissolving and adding lead acetate; now we scorify directly.

For the silver-assay, dissolve 0.5 A. T. of copper in 75 c.c. water and 50 c.c. nitric acid (sp. gr. 1.42); boil off nitrous fumes; dilute to 500 c.c.; add slight excess of hydrochloric acid and allow precipitate to settle on steam-bath over night. If precipitated hot, the silver chloride is curdy. Filter through 15 cm. No. 595 Schleicher and Schull filter. Wash into point of filter; roll up around precipitate; dry and char in front of muffle in 2½-inch Denver scorifier; introduce into forepart of

muffle and ignite at low heat; remove as soon as ignited; add 35 grammes test-lead and a pinch of borax; melt in hot and scorify at low heat; heat and pour. Button, 12 grammes. Cupel for "feathers." Deduct gold as found in gold assay.

For the gold-assay, the combination method was formerly conducted as follows: In each of two beakers, 1 A. T. of copper, 100 c.c. water, and 100 c.c. nitric acid, added in two portions. Boil off nitrous fumes; dilute with distilled water to 500 c.c., add 20 c.c. river water, 10 c.c. concentrated solution of lead acetate, and 5 c.c. concentrated sulphuric acid; settle over night cold; filter, char the filter-paper, and scorify as for silver. Cupellation-button, 15 grammes.

The gold-assay is now performed as follows: 1 A. T. is scorified in ten scorifiers, at 0.1 A. T. each. Three scorifications are given, and the ten buttons are combined for gold. The process is conducted as on matte. No corrections are made for losses in scorification or cupellation.

Results in Ounces Per Ton.

Borings.	Ag.	Au.
By scorification,	154.4	0.30
" chloride precipitation,	156.1	......
" combination,	......	0.25

As remarked above, the methods we have found most satisfactory are: On matte, scorification for silver, and crucible for gold; on copper, chloride without lead acetate for silver, and scorification for gold.

O.

Method A.—Dissolve 1 A. T. of matte or borings with nitric acid (diluted 1 : 1) in liter beaker, well covered, adding acid cautiously to avoid sudden and violent evolution of nitrous fumes. This is done cold until 200 c.c. of the dilute acid has been added; then add 20 c.c. of sulphuric acid and heat on sand bath till all copper is in solution, all nitrous fumes gone, and copper sulphate begins to precipitate. Cool somewhat; dilute to about 300 c.c., and filter through a double filter. This retains, in the case of matte, the gold and separated sulphur; in the case of borings, the gold and any insoluble residue. Wash filter, and make up filtrate and washings to about 700 c.c.; heat to boiling; add a slight excess of salt solution; stir; add 20

c.c. of solution of lead nitrate (1 : 5), and stir vigorously for five minutes. By this procedure the silver chloride is wholly entrapped in the precipitated lead sulphate, and the tendency to cloudiness in the filtrate removed. A crystal-clear filtrate is obtained if allowed to stand over night. After settling and standing, filter off and wash the lead sulphate and silver chloride; dry both precipitates in the air-bath; and we have in each assay two portions, the first containing all the gold with only a trace of silver, and the second the remaining silver as chloride, with large excess of lead sulphate. It should be remarked that C. P. nitric is used; so that if little silver is present to precipitate chlorine in the common acid no gold may be lost by its use.

The paper containing the gold is put in a 2½-inch scorifier and burnt off. As it carries but a trace of silver, no great care need be exercised except to avoid mechanical loss. In the case of mattes, allow the sulphur to burn briskly off, and then burn out all combustible matter.

The filter containing the lead and silver is now placed on top and burned at the *lowest possible* heat. Then cover with 30 grammes granulated lead; add but little borax-glass; scorify at low heat; and cupel the buttons with all precautions. Being free from copper, they behave well in the furnace. After weighing, part as usual for gold.

Method B.—This is the same as the foregoing, except that the

Results in Ounces per Ton.

Test.	Method A.		Method B.	
	Ag.	Au.	Ag.	Au.
Matte, 1	124.78	2.20	127.49	2.25
" 2	125.05	2.26	128.09	2.27
" 3	126.09	2.27	128.59	2.27
Matte, average	125.31	2.24	128.06	2.26
Borings, 1	156.90	0.21	159.55	0.22
" 2	157.06	0.22	159.33	0.22
" 3	156.94	0.22	160.14	0.22
Borings average	156.97	0.22	159.67	0.22

scorifier-slag and the cupel are powdered, fluxed, and assayed in a crucible, the silver and gold thus recovered being added to the results in method A.

In addition, an alternative method was tried on the matte, consisting in direct scorification of 0.1 A. T. portions with 60 grammes test-lead, adding borax. This method was employed on five portions, the buttons being combined, weighed, and parted. The results, without any allowance for slag- or cupel-absorption, were, in ounces per ton, silver, 127.44; gold, 2.26.

P.

Borings.—Dissolved 2½ A. T. in 500 c.c. nitric acid (15° B.), standing cold over night; clear solution decanted through filter; acid added to insoluble portion and heated until all copper was in solution. After standing till cold and clear, solution decanted through same filter and residue carefully washed on to filter. All solutions and washings combined; silver precipi tated with hydrochloric acid; settled until clear; chloride filtered, washed, and sprinkled with a little dry sodium carbonate. Both filters put together in scorifier with about 1 ounce pure lead, heated slowly to dryness and to burning of filter, and then to full scorification; cupelled, weighed, dissolved slowly in 15° B. nitric acid, and boiled for ten minutes, letting the matrass stand inverted for at least fifteen minutes. Gold annealed and weighed.

Matte.—Treatment practically the same as for borings, except that only 1 A. T. was taken and 22° B. acid was used.

Copper was determined in the matte only. Duplicate samples, 1 gramme each, dissolved in strong nitric acid and boiled until sulphur had fused to globules. Solution diluted; silver separated by hydrochloric acid; evaporated with sulphuric acid; copper separated by iron; redissolved in nitric acid, and titrated with potassium cyanide, with proof.

Results.

	Copper. Per cent.	Silver. Ounces per ton.	Gold. Ounces per ton.
Matte, ...	50.4	125.4*	2.1
"	50.7	124.4*	2.0
Borings, ...		156.74	0.502
"		156.70	0.50

* SECRETARY'S NOTE.—P. subsequently wrote that these figures did not include any allowance for loss in cupel, and that he desired to add 3.37 for the said loss,

Q.

Matte.—The matte was first assayed for copper by precipitating the copper on aluminum foil, and subsequently titrating with potassium cyanide. Four tests gave the percentage as follows: 54.17, 54.38, 54.41, 54.51; average, 54.37. Assay by electrolysis gave 54.6 per cent.

Assays of matte for silver and gold were made as follows:

Assay I.—Ten 2½-inch scorifiers, each charged with 0.05 A. T. matte, 0.05 A. T. powdered glass, 29 grammes test-lead (half mixed in, half as cover), all covered with 0.5 gramme borax-glass. Run hot for thirty-five minutes. Five of the buttons cupelled direct, and five rescorified with shells of slag which clung to them, together with ½ to ¾ gramme borax-glass, and a light cover of borax. Also run hot for thirty to thirty-five minutes. (*Note.*—Rescorified the slag from both scorifications, but could not find a trace of silver.) All ten of the buttons checked so closely that the difference between any two buttons could not be detected.

Weight, silver and gold in 1 A. T., 128.88 milligrammes. In parting, nitric acid of the strength of 2½ to 1 was used, followed by treatment with concentrated acid.

	Milligrammes.
Weight of gold in 1 A. T., ...	2.34
" " silver " "	126.54

Assay II.—Ten scorifiers, as above, except that charge consisted of 32 grammes test-lead, and nearly 0.1 A. T. powdered glass, covered with borax, and run cooler at first. Result showed, as in first assay, no difference between the single and double scorification, and no trace of silver in either slag. Moreover, this poured a trifle more fluid and looked rather better than the first assay.

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as determined by proof-assay under similar conditions. He says: "Our custom in practice has been to make up a proof, based on a preliminary assay, and run through at the same time, under the same conditions of heat, and with the same weight of lead as the assay in question. This was not done with the matte sample. It is our belief that where a preliminary assay is made, and the sample is cupelled between two proofs, the loss of the said proof, added to the weight of the sample-button, less the gold, will give a very trustworthy result, with the minimum expenditure of time and materials."

The figures given in the text should therefore be, for the two silver-assays of matte, 128.77 and 127.77 respectively; and the new average, 128.27, has been substituted in the summary on p. 253.—R. W. R.

Result: (Gold all parted as in first assay.)

	Milligrammes.
Weight of silver and gold in 1 A. T., ...	129.88
" " gold in 1 A. T., ...	2.34
" " silver in 1 A. T., ...	127.54

Assay III.—Ten scorifiers, as before, using as charge 0.05 A. T. matte and 0.05 A. T. raw unfused borax instead of glass, 30 grammes test-lead covered with one gramme borax-glass. None of the buttons were rescorified; slag tested for silver and none found.

Result: Eight buttons checked each other and two were a trifle light. (Weight of gold plus silver in 10 buttons averaged 128.3.)

	Milligrammes.
Weight of gold and silver, average 3 buttons, ...	128.70
" " gold in 1 A. T., ...	2.38
" " silver in 1 A. T., ...	126.32

Assay IV.—Ten scorifiers, as above; charge, 0.05 A. T. matte and 0.05 A. T. borax-glass, 34 grammes test-lead, with very light cover of borax-glass. Ran cool at first, then very hot. No rescorification; slag tested for silver and none found.

Result:

	Milligrammes.
Weight of gold and silver in 1 A. T., ...	130.08
" " " in 1 A. T., ...	2.38
" " silver in 1 A. T., ...	127.70

All buttons in this case checked fairly closely.

(*Note.*—In all of these four assays the acid used for parting was saved, neutralized with ammonia, and no trace of copper could be detected.)

Borings.—The borings were assayed for silver and gold as follows:

Assay I.—Ten 2½-inch scorifiers; 0.05 A. T. copper, 0.1 A. T. powdered glass, 29 grammes test-lead, covered with ½ to ¾ gramme borax-glass.

First 5 buttons cupelled; second 5 buttons rescorified with one gramme powdered glass mixed in; 28 grammes test-lead covered with ½ gramme borax-glass; scorified hot; 2 buttons lost in cupellation.

	Milligrammes.
Weight of gold in 1 A. T., ...	0.38
" " silver in 1 A. T., ...	157.02

Buttons from both single and double scorification checked very well, except one from the double scorification, which was about 0.1 light.

Acid used for parting, same as for copper matte, viz., 1 part concentrated nitric acid to $2\frac{1}{2}$ parts water, with final boiling in concentrated acid.

(*Note.*—No copper found in any of the silver solutions.)

Assay II.—Ten scorifiers taken; charge, 0.05 A. T. copper, 0.1 A. T. raw borax; 34 grammes test-lead, covered with borax-glass. Run very cool at first, ending up hot.

All buttons rescorified with 32 grammes test-lead, 1 gramme powdered borax-glass and $\frac{1}{2}$ gramme borax-glass for cover. Scorified hot. Six buttons checked exactly; four showed slight variations.

	Milligrammes.
Weight of gold in 1 A. T., ...	0.42
" " silver in 1 A. T., ...	156.78

Assay III.—(Wet assay in duplicate.) One-half A. T. borings, treated in beaker with 120 c.c. dilute nitric acid (1 acid, $2\frac{1}{2}$ water). Add 20 c.c. concentrated acid, after evaporating for 1 hour. When solution is complete, dilute, add strong solution of sodium chloride and allow to stand over night. Filter, dry precipitate, scorify with 35 grammes test-lead covered with inverted scorifiers, and run at very low heat at first. Remove cover and finish scorifications at comparatively low temperature. The buttons cupelled and parted as usual.

	Milligrammes.
Weight of gold in 1 A. T. (fire assay), ...	0.40
" " silver in 1 A. T. (original), ...	158.90
" " silver in 1 A. T. (duplicate), ...	159.40

The amount of gold deducted in above assay was determined by separate fire-assay. The silver buttons from three assays, when parted, yielded only a trace of gold. This was undoubtedly due to the fact that in precipitating the silver, the solution of salt was added in excess and chlorine set free, by which the gold was dissolved. In order to test this, a little gold (1 mgr.) was treated with a solution of acid and solution NaCl, and upon boiling readily dissolved.

Assay IV.—In order to avoid the loss of gold experienced in the third assay the same procedure was followed, with this exception: instead of filtering off the residue, after dissolving out the copper, add 1 to 2 drops of hydrochloric acid, to precipitate a portion of the silver on the gold; after which filter, add an excess of HCl to filtrate, and scorify both filters together. The results of this were as follows:

	Milligrammes.
Weight of gold in 1 A. T., ...	0.40
" " silver in 1 A. T. (original), ...	159.20
" " silver in 1 A. T. (duplicate), ...	159.60

The gold reported is from the scorified and cupelled precipitate.

(*Note.*—On all fire-assays, the slags were rescorified and no silver was found.)

R.

Method for both Matte and Borings.—One A. T. of sample was dissolved in dilute nitric acid, boiling out all red fumes. A small amount of silver was precipitated out by solution of salt, and filtered. Salt solution in slight excess was added to the filtrate, which, after settling over night, was filtered. The two filter-papers were combined; scorified at low heat; button cupelled and weighed, parted with nitric acid, and gold weighed. The method used to determine copper is not stated.

Results.

	Silver. Ounces per ton.	Gold. Ounces per ton.	Copper. Per cent.
Matte, I., ...	126.40	2.14	54.57
" II., ...	125.50	2.17	54.48
Borings, I., ...	149.47	0.26	97.46
" II., ...	148.10	0.26	97.59

S.

The matte sample was reground and passed through a 100-mesh screen; and it is urged that samples of such material should always be of uniform fineness, not coarser than that.

Determination of Copper in Matte. — Accurately weigh 1 gramme; brush into a 4-ounce lip-beaker; just cover with distilled water; add 10 c.c. concentrated nitric acid; cover quickly with watch-glass; heat till the nitric fumes form; then stand in a warm place until sulphur has clearly separated; carefully

wash down the watch-glass and sides of beaker; dilute to about 50 c.c. and bring to boiling. This will cause all sulphur to become white and flaky, and all copper and silver to be dissolved as nitrates. Filter through as small a filter paper as possible, to lessen absorbent surface and requisite wash-water. We have been using 8-ounce beakers for this filtrate, as best suited to our cylinders and apparatus. When filtrate has cooled, add slight excess of dilute ammonia (1 part distilled water to 1 part stronger water of ammonia), making sure that no free nitric acid remains; add through a burette dilute sulphuric acid (1 part of acid of sp. gr. 1.84 to 3 parts water) till the solution is just acid; and then 6 to 8 c.c. additional of the same (on account of the iron and zinc in the matte); cool; and the solution is ready for electrolysis, described below.

Determination of Copper in Borings.—Weigh 1 gramme, or thereabouts (any catch-weight up to 1.005 grammes will do, the object being to avoid multiplying any small error, and also to keep the electrolytes of several tests nearly equal); place *direct* into the beaker used for electrolysis; just cover with distilled water; add with a burette 4 c.c. concentrated nitric acid (we generally cover the beaker with a watch-glass, and, slipping back the glass a little, drop the acid down the beaker-lip, avoiding the spitting out otherwise likely, by reason of the violent reaction); let covered beaker stand cold until violent action has ceased, and nitrous fumes have almost disappeared; then heat, *not boil*, until all copper is dissolved, and the beaker is clear of brown fumes. Wash down watch-glass and sides of beaker, and proceed as for matte, except that filtering is omitted, and the final excess of dilute sulphuric acid added is only 4 c.c.

We think the solution thus obtained is the best electrolyte for such work, being a sulphuric solution (preventing any "burning" of the copper, and permitting easy current-adjustment), containing ammonium nitrate and sulphate (which prevent the precipitation of arsenic and antimony); completely holding iron as ferrous sulphate; and not as subject, as are nitric solutions, to decomposition or alteration. The copper deposit is of good color, and satin-like or crystalline in appearance. We use, by preference, cylinders slit on one side, permitting proper circulation of the liquid, without requiring the solution to be made to overflow the top of the cylinder. We dilute the electrolyte

until it comes within, say, 0.5 inch of the top of the cylinder, then start the electrolysis, usually in the evening, and let it run over night; in the morning, dilute to cover the entire cylinder, thus getting a test of the analysis. Before removing the cylinder, make sure that drops taken from the electrolyte show no coloration with sulphuretted hydrogen water. By removing a block from beneath the beaker, and disconnecting the anode spiral, the solution and spiral are quickly dropped and removed from under the cylinder, which is then detached, plunged into boiling water, and afterwards into alcohol, being thoroughly rinsed many times. Shake off drops of alcohol; ignite the cylinders over an alcohol lamp; cool in the ordinary temperature of the weigh-room, and weigh. The naked cylinders are similarly prepared for weighing before use. After deducting the weight of the cylinder, the remainder is that of the copper and silver; and the net determination for copper is obtained by deduct ing the percentage of silver. We prefer to precipitate copper and silver together in this way, experiments having satisfied us that attempts to separate the silver previous to electrolysis are untrustworthy, and dangerous to the accuracy of the copper determination. Moreover, they require filtration of the solution, which we contend should be avoided, if possible, in the case of copper alloy.

Assay of Copper Borings for Gold and Silver.—Weigh 1 A. T.; place in a liter-beaker; cover with enough cold water to prevent running over when acid is added; add 4 c.c. concentrated nitric acid for every gramme of copper taken; cover quickly; let stand cold under a hood till brown fumes are almost gone; then in a warm place till they are gone completely; wash down cover and beaker-sides; dilute with cold water till solution is cold (usually to 600 to 800 c.c.); add, from pipette or burette, solution of sodium chloride of known strength, in proportion to amount of silver contained; stir well; wash down sides; cover and let stand twelve hours at least. In material like this sample, the amount of silver chloride formed will carry down all the gold with it. For copper containing gold and only a small amount of silver, we would advise adding either a solution of some lead-salt and dilute sulphuric acid, or else an emulsion of lead sulphate, to help collect the chloride precipitate.

Decant the solution through a 12½ cm. Swedish paper of O

grade; wash precipitate from beaker upon filter; clean sides and bottom of beaker with a strong goose feather. When all chloride has been removed to the filter, and the wash water used in such removal has run through, wash the filter from the edges down with a very fine stream of absolute alcohol, which is allowed to drain through precipitate and filter paper as a dryer. We generally loosen the filter, sliding it up in the funnel; and it is soon dry enough to be wrapped, placed in the scorifier, and scorified at low heat. For scorifying, cover ash of filter, etc., with 1 to 2 grammes litharge, to prevent the carbon from causing spitting; add, in grammes, 25 to 30 c.p. granulated lead, upon that 1.5 to 2 fine ground silica and 0.5 borax. Pour carefully, to prevent "shotting;" cupel button (10 to 12 grammes) at a "feather heat;" brighten at a higher heat; and anneal before removing, to prevent "spitting." For parting, place assay-buttons in porcelain cups; cover with sufficient dilute nitric acid (1 part concentrated acid to 3 parts water) to dissolve the silver, and heat slowly and in best manner to keep the gold together, until all silver seems to have been converted into nitrate. Decant this solution from the gold with a glass rod; add about 5 c.c. concentrated nitric acid; heat and boil until solution is colorless; decant; wash gold in the cup twice with hot distilled water; decant completely; dry; anneal; cool and weigh upon an Oertling gold balance. We do not recommend the use of the gold balance for any other work.

Assay of Matte for Gold and Silver.—This may be done as above, by securing complete decomposition of the sulphides, and a white and flaky condition of the sulphur, before diluting for the addition of salt-solution. But the general method we use for matte is scorification and re-scorification, with heavy acid flux, to take up most of the copper. If carefully manipulated, this will give good results. Care must be taken not to have too much copper in the cupel button. When gold is to be determined, the buttons from several scorifications must be combined for parting.

Both the methods mentioned were employed on the samples of matte here under consideration, with practically identical results.

Results.

		Silver and Gold. Ounces per ton.	Gold. Ounces per ton.	Silver. Ounces per ton.	Copper and Silver. per cent.	Silver. per cent.	Copper. per cent.
Borings	I.,	159.2	0.28	......	97.92	......	......
"	II.,	159.2	0.28	......	97.92	......	......
"	III.,	159.3	0.29	......	97.92	......	......
"	Average,	159.3	0.28	159.0	97.92	0.55	97.37
Matte	I.,	128.8	......	......	55.52	......	......
"	II.,	128.4	2.27	......	55.52	......	......
"	III.,	129.0	......	......	55.50	......	......
"	IV.,	128.4	2.28	......	......	......	......
"	V.,	129.0	2.26	......	......	......	......
"	Average,	128.7	2.27	126.4	55.51	0.43	55.08

T.

For both matte and borings, gold and silver were determined by scorification and cupellation. Copper was determined by the cyanide method, the result thus indicated having been reduced to the basis of dry assay by deducting 1.3 per cent. These are the regular methods employed in this establishment.

Results.

	Gold. Ounces per ton.	Silver. Ounces per ton.	Copper. per cent.
Matte, . . .	2.33	130.42	50.75
Borings, . . .	0.37	160.63	97.98

Hysteromorphous Auriferous Deposits of the Tertiary and Cretaceous Periods in New Zealand.

BY HENRY A. GORDON, WELLINGTON, N. Z.

(Florida Meeting, March, 1895.)

UNDER the title "Hysteromorphous" it is proposed to include deposits which have been formed from original deposits by the influences of the surface-region.

The term Hysteromorphous—later-formed—has been used by Prof. Posepny, of Vienna, to distinguish secondary from primary deposits and is more comprehensively applicable than the terms or designations hitherto used for the same thing.

The terms débris, diluvium, drift, wash or gravel-deposit all convey the special idea of mechanical crushing, disintegration and reassortment to the exclusion of any chemical agency whatever. As it will be necessary to notice in this paper auriferous deposits of a secondary character which are probably, in part at least, chemical precipitations of gold from solution, the adoption of the term Hysteromorphous, covering, as it does, both modes of deposition, is considered desirable.

Chemical Deposits.

It is well known that gold, held in solution in water standing in old mine-workings, is often precipitated in a metallic state by the agency of organic matter, such as decaying timber, or by sulphides of other metals—iron, lead, etc. Such deposits are, however, generally unimportant; and, although instances are said to have been observed in New Zealand, they require verification.

In alluvial deposits it is claimed by many that at least a portion of the gold has been deposited by chemical agencies. A. R. C. Selwyn, Director of the Geological Survey of Canada, and F. A. Genth and Oscar Lieber, in the United States, have maintained the opinion that detrital gold generally, or a portion of it, has been deposited from solution. Lauri, J. A. Phillips, Wilkinson, Newberry, Daintree, Skey, Egleston and others have accepted this view as more or less generally applicable; while E. Cohen, Devereux and Posepny admit that a part of the alluvial gold in any field may be due to a chemical precipitation from solution, but, nevertheless, maintain that by far the greater part of such gold has been derived mechanically from auriferous lodes or from original altered or metamorphic rocks impregnated with gold.

No doubt solutions of gold exist in nature, and it is quite clear that solution and precipitation of the metal in the manner described by the various authors above mentioned may and do take place; and it is purely a question of favorable or unfavorable conditions which mode of deposition shall predominate. All who have given attention to the subject, however, admit that the deposition of gold in the alluvial drift in New Zealand is due to mechanical action to a far greater extent than to any other means. Of all the Australian colonies and

most of the gold-producing countries in the world, New Zealand presents the least favorable conditions for the solution and precipitation of gold in hysteromorphous formations. Its rainfall is considerable, the temperature is not high and the denudation of the higher regions is comparatively rapid. The evaporation is less than in the other Australian colonies, and, consequently, the rivers carry to the sea a large percentage of the total rainfall, and with it the greater part of any gold which may be in solution.

In the other Australian colonies and in countries having similar climatic conditions the results may be different. Although the rainfall during the year may equal, or even exceed that of New Zealand, and the underground circulation be sufficient to dissolve and carry forward such mineral salts as may be formed, it is evident that where, by reason of excessive evaporation, only a small proportion of the rainfall reaches the sea, a much greater precipitation of gold might take place than under the conditions prevailing in New Zealand. It can scarcely be doubted that in countries having the climate of Queensland or Western Australia the quantity of gold thus precipitated must be greater than in New Zealand; yet such deposits are not absent from the gold-bearing drift of this country. As might be expected, they are found chiefly in the dry district of Otago. Crystals of gold have been found in Mahakipawa creek, Marlborough. One very beautiful example, an octahedron, came into the possession of Dr. Keyworth, of Wellington.

In central Otago, where the climate is comparatively dry for the greater part of the year, an unquestionable precipitation of gold occurs. In the Upper Taieri district, "wire-gold" is occasionally met with in considerable quantities. It is always obtained at or near the surface, and was in the early days of the diggings considered by many of the miners to be petrified grass-roots. This form of gold is now less frequently met with; but in 1881 McKay, the mining geologist, saw a parcel of 40 ounces of gold of this character at Naseby, and purchased a few pennyweights of it, comprising some remarkable pieces, from the bank where it was sold. There were thin wires of gold straight or bent, one side of which was smooth or striated, the other side being covered with small cubical crystals of gold.

This sample, or part of it, was taken to Sydney and exhibited at the first meeting of the Australian Association for the Advancement of Science.

Last year Mr. McKay saw, in the possession of Mr. John McKersie, a fine specimen of the same character of gold, which was obtained somewhere in the neighborhood of the Carrick range. It was about 1½ inches in length and about 2 inches in diameter, and, like the specimens already mentioned, smooth on one side and covered with crystals on the other. Such gold might possibly be derived from the denudation of lodes; but the probabilities are against this view. It is to be noted, that the samples hitherto found came from districts where the quartz drift deposits of Cretaceo-Tertiary age are auriferous; and as the strata are tilted to considerable angles, gold-bearing bands are thus exposed at the surface, where a partial solution of the gold might be again precipitated by organic matter and crystallized at the outcrops.

Another possible instance of gold precipitated from solution is reported in the Maerewhenua field; but the occurrence requires verification. Last year Mr. McKay thought he detected minute crystals of gold which were supposed to be derived from a greensand band on that field. Samples of the greensand subsequently sent to Wellington, yielded no gold; but the formation was declared by the miners to be gold-bearing. The greensand is a marine deposit, full of shark's teeth and shells of various mollusks, and consists of glauconite and very fine quartz sand, that has evidently been deposited in comparatively deep and still water, and is therefore very unlikely to contain mechanical gold, however fine.

The underlying quartz drift has auriferous layers or bands through it, and extends over a very large area of Otago. Sloping as it does from higher to lower ground, as on the flanks of the Kakanui range, it would at the lower levels become saturated with water, possibly chlorinated water, which would dissolve a part of the gold in the lower drift, and, under pressure, would rise into and saturate the greensand band, while the organic matter in this band would be sufficient to effect the precipitation of the gold now present in the greensands.

Mechanical Deposits.

Recent and Pleistocene gold deposits need not be adverted to in this paper, and, as many of the later pliocene gravel deposits are not readily distinguishable from those of still younger date, the most modern deposits that will be mentioned here are probably of older Pliocene date.

Pliocene and Upper Miocene Deposits of the West Coast of the South Island.

The youngest of these are the gravels of the Humphrey's Gully range, on the southern side of the Arahura valley. Similar gravels are noticeable at O'Donohue's creek, 5 miles from Kumara, on the Kumara-Christchurch road. At the latter locality, though gold is present, and the bed of the creek has been worked, the gravels indicated have not as yet proved profitable. At Humphrey's Gully, gold-mining has been carried on in these gravels for a long series of years, and there is yet an unlimited supply of material to be operated upon. Auriferous gravels, it may be of a slightly greater age, are worked in the Totara district on the top of Mount Greenland, 3000 feet above the level of the sea; nearer the sea-level in the Mount D'Or claim near Ross, and in other claims in and along the eastern margin of Ross flat; while on Ross flat the same gravels are found 240 feet below sea-level and contain large deposits of gold.

The same beds have a very large development in the north-western part of Westland, but, owing to the small percentage of gold yielded, they are not worked at the present time. They also have a large development along the east side of the Grey valley, more especially in the valley of the Little Grey; and at the source of this stream they fill the valley between the Paparoa range and the hilly country to the southeast of Reefton.

In the Inangahua valley they have an important development between the north branch of the Inangahua river and Boatman's and Larry's creeks; in the southwest part of the upper Buller valley their distribution has been less definitely ascertained; but between the junction of the Owen and the outlets of Lakes Rotoroa and Rotoiti, north of the Devil's Grip, an

enormous development of these gravels fills the northern slope, between the mountains to the east and west, to the shores of Blind Bay.

The same gravels are largely developed along the east coast of the South Island, but neither in the Marlborough nor in the Canterbury district are they known to be gold-bearing.

In the interior district of Otago they have again a large development, and are mined for gold at the Kyeburn, on the eastern side of the Maniototo plain. They are largely developed in Ida valley and in the Manuherikia valley, and appear at several places in the Molyneux valley, below the junction of the Manuherikia river.

In the Upper Clutha valley they have considerable developments at Bannockburn, along the Lindis, in the Cardrona valley, etc., and in these localities have been or are now mined for gold. Finally, at Switzer's, in the Waikaia-Mataura watershed, they have a considerable development, and are important as gold-bearing strata.

Beds of the same age cover considerable areas of the North Island, but are not known to be, and it is unlikely that in this part they are, gold-bearing.

The beds at most places have a moderate inclination to the horizon; at many places the dip is high; and at a few places they are standing vertical, in some instances inverted and thrown back upon themselves—as, for instance, at Tinkers, in the Manuherikia valley, and on Criffel Face, in the Cadrona valley.

Lower Cretaceo-Tertiary Deposits.

On the west coast of the South Island auriferous deposits of this age are, so far as is yet known, confined to the Nelson district. At Cement Town, Murray creek, and at Painkiller near Reefton, also in the upper part of Lankey's Gully, and at different parts of the range between the two branches of the Inangahua as far as Garvie's creek, the quartz-cements at the base of the coal-bearing series are known to be auriferous.

Gold is also reported as found in the coarse granitic conglomerates at the base of the Coal-Measures in the Ten Mile creek, north of Greymouth; and there is a probability of gold being found more or less in the same horizon between the Paparoa

range and the sea to Cape Foulwind, and the gorge of the Buller river.*

Gold occurs on the table-land west of Mount Arthur, in quartz-drifts occurring at the base of the Cretaceo-Tertiary formation, as developed at that place; and there is a large development of the same grits, probably auriferous, between the Karamea Bend, the upper Karamea, and the upper valley of the Mokihinui. In the upper Takaka valley there is also a large development of quartz-drift, known to be auriferous, underlying the limestones of Cretaceo-Tertiary date; and in the Collingwood district the first discovery of alluvial gold was made in white quartz-gravels of this age. At Golden Gully, in the Aorere valley, the quartz-drifts of Crètaceo-Tertiary date, or a re-wash of them, still resting on the original grits, was enormously rich, and yielded in a very short period (a few months) to a comparative handful of inexperienced miners, £40,000 worth of gold. In late years attempts have been made to work the deep ground at this locality, and at various places along a line of supposed fault, by way of the upper part of Appos creek and the Glenmutchkin Gully across the Parapara to the Glengyle claim, and thence across the ridge of hills to the flats at the head of Parapara Inlet. There can be no doubt as to the existence of gold along this line; but with the increasing depth and the difficulty of working the ground, there does not seem to be a corresponding increase in the amount of gold present; and there is little likelihood that the rich finds of the early days will be rivalled by what may be won in a few weeks of the future.

Nevertheless, if we are to be guided by report, there is yet an immense quantity of gold held within special small areas along the line indicated. Sir James Hector estimates the amount of gold that may probably be won at values ranging from £295,250 to £5,905,000 (*vide* his *Geological Report*, 1890–91, page 12). It is not always safe to make estimates of the amount of gold in ground that has not been thoroughly tested, but the above is sufficient to show that there is a fair prospect of a considerable amount of gold being obtained.

* It is found also over a large area in the upper Buller valley, where important developments in these beds are likely to take place. Though gold is known to occur in them, they have been but little prospected hitherto.

On the east coast of the South Island, the first (most northerly) auriferous deposits of this age are met with in the Kakahu, a branch of the Arawhenua or Temuka river. In the Kakahu, the auriferous bed lies at the base of the coal-bearing series, and consists of rusty-colored quartz-cements that have yielded gold at the rate of 1 oz. 5 dwts. per ton. No regular workings for gold have been carried on at this place. Within Otago, in the Waitaki valley, the bulk, in fact almost the whole, of the gold obtained from the Maerewhenua gold-field has been obtained from the lower beds of the Cretaceo-Tertiary formation; and the quartz-drifts of this field will probably be found auriferous for some distance to the southeast of the Kakanui gorge.

Further to the south and southeast in the Horse range there is a great accumulation of sub-angular breccia-conglomerates well displayed in the gorge of Trotter's creek and on the southwest slopes of the Horse range. These deposits lie at the base of the series forming the Coal-Measures of Shag Point. The breccia-conglomerates are auriferous, and have yielded to the creek-gravels on both sides of the range enough gold to make the working of them pay well in places. From this source, or from beds of the same age, comes the gold found on Moeraki beach and the several sea-beaches worked farther to the north as far as Oamaru.

At almost all the important diggings around the borders of the Maniototo basin the gold is obtained from quartz-gravel of old Tertiary or Cretaceo-Tertiary date or from a rewash of such mixed with modern creek-gravel from the adjoining ranges. This also is the case in the Ida valley, along the whole length of the Manuherikia valley, from St. Bathaas to Clyde, and to some extent also at Bannockburn, in the Clutha valley above Cromwell, and in the Nevis valley. On Mount Buster, north of the Maniototo plain, the quartz-drifts of Cretaceo-Tertiary or Cretaceous date reach a height of 4000 feet above sea-level; west of this they mostly occupy much lower levels, being, in the neighborhood of Clyde, little more than 400 feet above tide; but on the west side of the Clutha valley, again on Mount Criffel, and a third time at the source of Skipper's creek, they reach an altitude close upon 4000 feet. These old quartz-drifts occur at many places in the middle part of the Taieri valley, and

stretch along the coastal tract to the mouth of the Molyneux. In the Tuapeka district both the breccia-conglomerates and the quartz-drifts are auriferous, the celebrated Blue Spur deposit* belonging to the breccia-conglomerate beds of this age.

Quartz-drifts of this age, but sometimes said to be of Miocene age, are, west of the Molyneux, largely developed along the lower Pomahaka river and in the country west of the Tapanui mountains to the Mataura river, the most notable diggings being Waikaka and Switzer's. At the first-mentioned place all the gold has been derived from strata of quartz-drift and white clay, which, in a nearly vertical position, have been denuded to form the gravels of those creek-valleys which have proved rich in gold.

At Switzer's the quartz-drifts have been worked at several places, and are still worked by Chinamen.

Beyond the Mataura, in the Southland district, the quartz-drifts form an extensive area to the south and west of the eastern part of the Hokonui hills, where the gravels skirt round the southeast end of the range. They are known to be auriferous; but farther to the west and northwest, though the gravels are known to occur abundantly, it has yet to be shown whether they are auriferous or not.

Jurassic and Pre-Jurassic Deposits.

There are several bands of coarse conglomerates, some of them of great thickness, in the middle and old secondary strata of the Otago, Southland and Nelson districts. Some of these have been reported as gold-bearing, but so far no locality has been discovered where gold is found in paying quantity, and therefore these may be disregarded at the present time.

There is in the Nelson district, at the source of the Baton river, a conglomerate composed almost wholly of vein-quartz. It occurs in the Te Anau series, of Devonian age, and indicates that although most of the conglomerates of Middle and Old Secondary age are chiefly of granite, the reason is not that no quartz, as vein-quartz or as folia in metamorphic rocks, existed at the surface. At the time of their formation, however, the great schistose tract of Central Otago was probably.covered by

* Described by T. A. Rickard, *Trans.*, xxi., 432, 445.

Carboniferous and Permian deposits. The quartz-drifts of Cretaceo-Tertiary date have been formed, for the most part, by the action of the sea, from the material of the breccia-conglomerate or materials of like origin. In Central Otago the auriferous quartz-drifts are mainly lacustrine, and in age they may be somewhat younger than the similar deposits to the east and south, or in the northwestern part of the district from Lake Wakatipu to the Upper Shotover.

The Equipment of Mining and Metallurgical Laboratories.*

BY H. O. HOFMAN, ASSOCIATE PROFESSOR OF MINING AND METALLURGY IN THE MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(Florida Meeting, March, 1895.)

THE mining and metallurgical laboratory, as we understand the term in this country, is a place in which mechanical and chemical working-tests are made on ores, fuels and furnace-materials. It is of quite recent origin. The first laboratory of this kind to be used in connection with teaching was put into operation in 1871 at the Massachusetts Institute of Technology.† The idea had already existed in the mind of President W. B. Rogers when he wrote, in 1864, his pamphlet on "The Scope and Plan of the School of Industrial Science of the Massachusetts Institute of Technology;" but several years elapsed, and an extended visit to the mines and mills of Colorado, Utah, Nevada and California was required before this idea could take a form adapted to the purposes of original research as well as of instruction. The laboratory was given from the first into the charge of Prof. R. H. Richards, who, by improving its methods and enlarging its scope, has brought it to the position which it occupies to-day as the leading representative of its class. Private laboratories for making tests upon ores had previously existed here and there, especially on the Pacific coast, for silver and gold ores; but in the educational field the

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* SECRETARY'S NOTE.—By consent of the Council, this paper is presented also to the Society for the Promotion of Engineering Education.

† R. H. Richards, *Trans.*, i., 400.

Massachusetts Institute of Technology was the pioneer. To-day there is hardly a school of mines in this country that has not a more or less complete mining and metallurgical laboratory. In European mining-schools there is very little laboratory-teaching. Most of them are located in mining-districts, where the students can personally see and engage in the practical work of mining, concentrating and smelting. Those which are in large cities, at a distance from mines, labor under a great disadvantage. The student only sees practical work when he makes an occasional visit to mining regions, and is otherwise left entirely to theory. It must not be inferred, however, that the location of a school in a mining district can make the laboratory superfluous. On the contrary, one who, like the present writer, has received his training in such a school, sees clearly afterwards how one-sided becomes the teaching in a mining district without the addition of such laboratory-work. The instructor is only too liable to give most, if not all, of his time to elaborating unnecessary details of the local methods, past as well as present, and to pass over with amazing celerity those branches of the subject not represented in his district. Yet even as regards local work, upon which he puts such undue stress, he is likely to be too theoretical, because, not being practically engaged in it, or able to apply such tests as are furnished in the laboratory, he necessarily falls into too abstract a way of viewing the whole subject. The result is that his instruction tends to produce theorists, who speak with unwarranted assurance concerning the most difficult problems which the engineer has to solve; but who, if confronted with a simple, concrete question, are at a loss what to do.

That this lack of laboratory-training in German technical schools (which are among the foremost in Europe) is beginning to be realized as a defect was evidenced by the intense interest and careful study bestowed upon the subject by the commissioners who came to the Columbian Exposition two years ago. They did not hesitate to praise our system and to express the hope that it might be adapted to meet their necessities on the other side of the Atlantic.

The mining and metallurgical laboratory, then, as developed in this country, may be considered a necessary adjunct to every school of mining engineering. In it the lecture-instruction is

illustrated with practical experiments, carried out by the students themselves. But it has also a larger scope. By the method of experiment, the student learns how to take hold of each problem as it presents itself and carry it through the different stages until it is, or the reason is discovered why it cannot be, satisfactorily solved. He is thus taught to observe closely, to make careful notes, to compare the results obtained and draw his own inferences and conclusions, and, finally, to report what he has done in clear and accurate language.

In fitting up a laboratory, we have to consider only the departments of mechanical concentration and metallurgy. Practical mining can be taught only in the mine. Some schools (for instance, the one at Ballarat, Victoria, Australia) are provided with a model of full natural size, showing a shaft with the lode, cross-cuts, etc. While this, apart from the question of expense, is an improvement on the small models formerly so extensively found at schools, it cannot but give a false impression of what a mine really is. The practical study of mining, in this country at least, is carried on to-day in "summer schools." The students spend some time in mines, going systematically through the different kinds of work, and thus becoming sufficiently familiar with mine-operations to listen understandingly to lectures on the subject. It is the merit of Prof. H. S. Munroe, of Columbia College, to have given to the summer school of mining such an impetus that to-day there is hardly an American mining school without this auxiliary course.

Before discussing in detail the equipment of a laboratory, it is desirable to consider the relation which the laboratory-plant should bear, as regards general arrangement and the kind and size of apparatus, to the large-scale working-plant of actual practice. A commercial concentrating-works, for example, must treat daily a considerable quantity of ore, and must work cheaply, which can only be done if the machines are so connected with one another that the ore shall receive a minimum amount of handling after the work is once under way. In the laboratory, on the other hand, the work, being purely experimental, must be carried on, step by step, in a deliberate and tentative way; and it is therefore essential that the operator shall be able to inspect the material under treatment before and

after every operation. Consequently, the machines must be separate, that they may be easily accessible for starting, stopping, accelerating and retarding, and may be connected at will; in short, that the work may be modified indefinitely under the immediate eye of the experimenter. A laboratory in which this principle is neglected carries in it the germ of failure. The writer was once connected with such an establishment, in which a full-sized ore-dressing plant had been erected according to the plan followed in commercial work, viz., the crushed ore was raised by a bucket-elevator to a set of screens placed in a line step-wise, one discharging into the other, and the sized products falling directly upon the jigs and the table below. Of course, a few tons of ore were quickly disposed of; but when the products obtained were examined after the experiment, the observer did not know very much more than he had known before. Such a working-plant may be of some value for obtaining more accurate quantitative results after all the necessary details have been determined by the use of detached machines; but it will do little more than substantiate what has already been sufficiently proven.

There are two opposite views concerning the kind and size of machinery proper for laboratory-use. One holds that it should follow as closely as possible that of a working plant. The other maintains the superiority of somewhat different and smaller apparatus as better suited to experimental purposes and also more economical. Having tried both kinds, the writer decidedly prefers the latter, especially for educational purposes, and is of the opinion that there are few mechanical questions to which a machine smaller than the commercial size cannot give a satisfactory answer. In addition to economy, convenience and other considerations, the saving of physical strain upon the student secured by the smaller apparatus is of importance. Fatiguing operations, especially for those unaccustomed to the work, exhaust the powers and unfit the student for mental effort.

The best size for the single machine can only be arrived at by repeated trials, which have now been made for almost all given cases, as will be shown later on.

In the discussion of the details of a laboratory, it will be more profitable to start from the basis of an actual working

laboratory, whatever may be its defects, than from an imaginary perfect one. The laboratories of the Massachusetts Institute of Technology, shown in plan in Fig. 1, may well serve this purpose.

The following are the different rooms, pieces of apparatus, etc., referred to by numbers in Fig. 1. In the present paper numbers enclosed in brackets are to be understood as referring to this figure.

1. Milling-room.
2. Blake Challenge rock-breaker.
3. Cornish rolls.
4. Gates rock-breaker.
5. Hendrie-Bolthoff sample-grinder.
6. Iron sampling-floor.
7. Cornish feeder.
8. Automatic feed-trough.
9. Richards *Spitzlutte.*
10. Coarse Collom jig.
11. Fine Collom jig.
12. Convex continuous round table.
13. Hendy Improved Challenge ore-feeder.
14. Stamp-battery.
15. Amalgamated plates.
16. Frue vanner.
17. Richards movable sieve-jig.
18. Water-tanks.
19. Steam-drying tables.
20. Bucking plates and Taylor hand-crusher.
21. Sampling-table.
22. Ore-bins.
23. Pounding-block.
24. Upright engine.

24.1. Morrel agate mortars.

25. Dynamo, 50 V by 50 A.
26. Dynamo, 2 V by 50 A.

26.1. Revolving barrel.

27. Depositing-table.
28. Leaching-tubs.
29. Larger amalgamating-pans.
30. Small amalgamating-pans.
31. Settler.
32. Tank.
33. Space to grow in.
34. Store-room.
35. Blacksmith's drilling-machine.
36. Carpenter's bench.

36.1. Ball-mill.

37. Assay-room.
38. Students' desks.
39. Pulp-balances.
40. Muffle-furnaces.
41. Crucible-furnaces.
42. Stack.
43. Iron table.
44. Balance-room.
45. Button-balances.
46. Store-room.
47. Store-room.
48. Furnace-room.
49. Blacksmith's forge.
50. Anvil.
51. Blacksmith's table.
52. Water-jacket blast-furnace.
53. Furnace ore-bins.
54. Brückner roasting-cylinder.
55. Copper-refining furnace.
56. Large hand-roasting reverberatory.
57. Roasting-stall.
58. Cast-iron kettle.
59. Larger cupelling-furnace.
60. Small hand-roasting reverberatory.
61. Small cupelling-furnace.
62. Pot-furnaces.
63. Space to grow in.
64. Professors' laboratory.
65. Table for electrolytic work.
66. Experimental *Spitzlutte.*
67. Chemical desks.
68. Hood.
69. Blow-pipe room.
70. Tables.
71. Cases for apparatus, etc.
72. Sink.
73. Library.
74. Book-cases.
75. Space to grow in.
76. Table.
77. Professors' desks.

78. Lithographic notes, etc.
79. Toilet-room.
80. Lockers.
81. Basins.
82. Closets.
83. Professors' room.
84. Stack.

These laboratories are located in the basement of the Rogers building, in the main building of the Institute, and comprise the entire department of mining-engineering and metallurgy, with the exception of the lecture-rooms and collections. While at first* all the metallurgical work, including dry-assaying, was done in the room marked [48] and the milling-work in the space now covered by machines [13] and [16], there are to-day a separate furnace-room [48], an assay- and balance-room [37, 44], a milling-room [1] and a blow-pipe room [69]. To these may be added two storage-rooms [46, 47], a toilet-room [79], a library [73] and the private laboratory [64] and office [77]. Upon closer inspection, it will be seen that the apparatus is pretty closely crowded. Although there is some "space to grow" [33, 63, 75], and there are places near [1] and [33] still open, there is little room for additional permanent machinery, the available space being necessary for erecting temporary apparatus and giving room to move about in. A laboratory built to-day with a liberal allowance of space and of funds would probably be planned somewhat differently as regards general arrangement, and would also possess a larger amount and variety of apparatus. The work in it would be easier and could be more conveniently and quickly, but not better, done.

In discussing the machines and furnaces, sufficient data will be given to enable the reader to form a clear idea of the relation which the laboratory-apparatus bears to that used in large-scale work.

The apparatus of the laboratory is best classed under three heads, corresponding with its purposes:

A.—Concentrating.

B.—Sampling and assaying.

C.—Metallurgical.

A.—Concentrating Apparatus.

1. *Coarse Crushing.*—Coarse-crushing is represented by the

* R. W. Raymond, *Statistics of Mines and Mining*, 1874, pp. 499 and 500.

Blake Challenge rock-breaker [2], with a receiving-capacity of $4\frac{1}{2}$ by 5 inches, and the Gates rock-breaker [4], with a receiving-hopper 12 inches in diameter. The machines are at a sufficient height above the platform to allow a wheelbarrow or bucket to be placed below the discharge. A pipe, connected with a small suction-fan, serves to carry off the dust, if desirable. The Blake is used for crushing lump-ore, the jaws being set $1\frac{1}{4}$ inches apart; the Gates for smaller sizes, the liners being set at $\frac{1}{2}$ inch. The Dodge and Lowry crushers may be added to the plant if it is desired to crush ore more uniformly than can be done with the Blake or the Gates type; but this will hardly be necessary for the testing of ores, although it might be useful for illustrating class-work. The small Taylor hand-crusher [20] is very convenient for breaking up specimens.

2. *Fine-Crushing.*—For fine-crushing there are: a pair of Cornish rolls, a stamp-battery, a non-discharging ball-mill, sets of pans, a sample-grinder, and bucking-plates.

The Cornish rolls [3], 9 inches in diameter and 9 inches in face, are of chilled iron, without the outside shell so common for large-scale work; are driven by direct- and cross-belt, and make 70 revolutions per minute. The pressure on the sliding-box is maintained by springs. The rolls have a large feed-hopper, with adjustable discharge-slot, holding about 100 pounds of quartzose ore. The crushed ore is directed by three converging pieces of sheet-iron (a short, steep one at the back, and a long, flatter one on each side) towards an oblong opening, $5\frac{1}{2}$ by 27 inches, through which it drops into an oblong sheet-iron box, 14 by 36 inches, of No. 22 iron, with sides 6 inches and ends 4 inches deep. The upper edges of all sheet-iron boxes or vessels used in the laboratory are bent around a $\frac{1}{4}$-inch iron rod to give them strength, and are painted with asphalt-varnish. If the ore is to be screened, an oblong wooden screen-frame, 54 by 11 inches inside dimensions, made of $2\frac{1}{2}$- by $\frac{7}{8}$-inch wood, and closed at the upper end, is suspended in a slightly inclined position from four iron ($\frac{5}{16}$-inch) hooks from the wooden frame of the rolls, and oscillated by an excentric of 1-inch throw and 200 shakes per minute, driven from the main shaft below. The ore drops upon a piece of sheet-iron, 11 by 12 inches, in the upper end of the frame, passing over which it comes to the screen (54 by $12\frac{3}{4}$ inches). Through this the finer

parts fall into a sheet-iron box, while the coarser ones are carried over into another which adjoins the first. The screens are fastened to the lower sides of their frames by means of angle-hoop-iron and screws.

The crushing-capacity of the rolls per hour is 600 pounds of quartzose ore to $\frac{1}{4}$-inch size, or 300 pounds to $\frac{1}{8}$-inch, or 150 pounds to $\frac{1}{16}$-inch. While they serve their purpose for fine-crushing, as a preliminary operation in ore-dressing, yet, if ore is to be rolled previous to chloridizing and leaching, Krom rolls are very desirable for finishing, the Cornish rolls serving in that case as roughing-rolls.

Roller-mills, such as the Huntington, Griffin, and Tustin, or discharging ball-mills, such as the Brückner, while doing satisfactory work in dry- and wet-rolling, are better suited for the mill than the laboratory, on account of the difficulty of cleaning up.

The stamp-battery [14 and Fig. 2] is of the California pattern. It has the usual single-discharge mortar for wet-crushing, but only three stamps; the weight of the stamps is 228 pounds; the mortar-bottom is $19\frac{3}{4}$ by 6 inches; the depth 5 inches; the discharge surface 20 by $10\frac{1}{2}$ inches; the screen-frame $21\frac{1}{2}$ by 13 inches; and the screen-surface $18\frac{7}{8}$ by $9\frac{1}{4}$ inches. The cams permit the lifting of the stamps to a height of 8 inches. The rate of crushing Nova Scotia gold-quartz with a 7-inch height of discharge, a length of drop of $5\frac{1}{4}$ inches and 98 drops per minute is 3353 pounds in twenty-four hours, or 1 pound for every 4198 foot-pounds developed. With a $7\frac{1}{2}$-inch drop and 60 drops per minute, it is 2117 pounds, or 1 pound for every 5816 foot-pounds. The coarsely-crushed ore is fed to the battery by a Hendy Improved Challenge Ore-Feeder [13]. A double-discharge mortar, of which one side can be closed by an iron plate, will soon replace the old mortar, so that in the laboratory it will be possible to do both dry- and wet-stamping. In planning a new mill a battery with three stamps would not be chosen. The choice would lie between a 5-stamp battery of light stamps, say 300 pounds each, a 1- or 2-stamp battery, the stamp weighing 750 pounds, and a steam stamp. The 5-stamp battery has the advantage that the same number of stamps is used as in common practice. It would not be feasible to have a full size 5-stamp battery, as it entails too much work and

requires more ore than is convenient and suitable for experimental work in the laboratory. The 1- or 2-stamp battery with 750-pound stamps dropping in a narrow double-discharge mortar, one side of which could be closed at will, the discharge to be on a level with the base of the die and to be raised by chuck-blocks to 16 inches, and the stamps to have a length of drop of from 4 to 10 inches, would be very acceptable. The results obtained with it would resemble very closely those of large scale work. As to the desirability of a steam-stamp for laboratory use, the writer feels himself at present unable to express an opinion.

The other fine-crushing apparatus, such as the ball-mill, the pan, the sample-grinder, the bucking plate, etc., will be discussed under the heads of sampling and metallurgical apparatus.

3. *Sizing.*—The sizing or sifting of ore is more tedious in the laboratory than it is in the mill, because the screening surface is necessarily smaller, and all sifting has to be done without the use of water. If there is only a moderate quantity of ore, the sizing is best done by hand on a platform covered with an iron plate [6]. Sieves with wooden frames from 24 to 18 inches in diameter, and iron or brass wire-gauze having from 4 to 20 meshes to the linear inch, are well suited for this purpose. With very small quantities of ore, nests of sieves with metal frames, 8 inches in diameter, and wire-gauze ranging from 20- to 120-mesh are convenient; the screenings to be caught in a metal pan. With large quantities of ore the sifting has to be done by machinery, and the shaking sieves referred to above, are used for this purpose. There are fourteen of these, representing the sizes 2-, 4-, 5-, 6-, 8-, 10-, 12-, 16-, 20-, 30-, 40-, 50-, 60- and 80-mesh. They sift per hour about 2000 pounds of 8-mesh ore, 1000 pounds of ore ranging from 14- to 30-mesh, 300 pounds of 50-mesh, and about 150 pounds of 60- to 80-mesh material. As this work is somewhat slow, it is better to do it in separate sizing-boxes. Two inclined boxes having screens of 3-, 10-, 18-, 30- and 60-mesh, and 4-, 8-, 14-, 24- and 50-mesh respectively, are satisfactory for the purpose. They are made of ½-inch pine, are 90 inches long, 18 inches wide and 5 inches deep, and have wooden covers screwed down on a felt band. They are oscillated 200 times per minute by an eccentric and connecting-rod, which gives them an end-shake. The

ore is fed into the hopper at the upper end, and drops on a piece of galvanized-iron, whence it passes on to the first (the coarsest) sieve. What is too coarse to pass strikes a dam at the opposite end and is discharged into a vertical spout at the side, to which a cloth bag is attached, through which it passes into a pail. It would seem as if the Coxe gyrating screen, which does such excellent work in sizing all sorts of minerals, might well be suited for laboratory purposes, either in the form of a single screen or a nest of screens. The trommels, as commonly employed in large scale working plants, are out of place in a laboratory. If a trommel is to be used, the polygonal form seems the most suitable, as the different screens could be easily adjusted and removed. It would be necessary in all cases to house the trommel.

4. *Hydraulic Classification.*—Hydraulic grading is done at present in the Institute laboratory only in an ascending current of water. Grading in a horizontal current of water, or *Spitzkasten*, will shortly be introduced, as it has been proved to be indispensable for the successful working-up of fine slimes. Now, the fine sands and slimes are only settled, but not graded. Hydraulic classification is practiced with small samples of finely-pulverized ore, as a preliminary test before working small lots. The samples are treated in the Richards pointed tube,* where the mixed sands, held in equilibrium by an ascending stream of water are, by slightly slackening the current, drawn off slowly into the glass bulb, which, when filled, is exchanged for another. The contents of each bulb are then separately sifted through a nest of graded sieves, and weighed and examined, to find out just how effective the work has been, and what will be the best sieve-size for the trial-test. In working, the material, after it has been crushed to the proper size, is passed through the automatic feed-trough [8], or the Cornish feeder [7], into a Richards *Spitzlutte* [9], when the discharge of the spigot will go to the jigs [10 and 11] and the overflow either to the vanner [16] or the slime-table [12], or first to the former, and, as tailings, to the latter. It is proposed to have the overflow, when worked directly on the slime-table, run first over a *Spitzkasten*, and then to feed separately

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* *Trans.*, xxiv., 430.

the spigot-discharge, thus insuring better work. Another way of using the Richards *Spitzlutte* is to feed only carefully-sized ore, when the spigot, in many cases, will give clean heads and the overflow clean tailings, provided there are no included grains. The capacity of the *Spitzlutte* with a $\frac{1}{2}$-inch spigot, is about three-quarters of a ton of sized material to 1 ton of mixed material per hour.

The automatic feed-trough and the Cornish feeder serve to convert dry pulverized ore into liquid pulp, delivering it to the *Spitzlutte*, the jigs or the slime-washers. The feed-trough is of wrought-iron, 10 inches wide at the top, 3 inches at the bottom and 7 feet long, and is placed in an inclined position on a wooden trestle. On the inner side the trough is marked off, so that the same quantity of ore may be washed down by the travelling jet in the same interval of time, which is usually one minute. The travelling jet is a $\frac{3}{4}$-inch iron pipe, pointed downward and fixed in a wooden truck, having two of its wheels on one edge of the trough and the other on a rail 3 inches away from the opposite edge. The pipe is connected by a rubber hose with the water-main. The carriage is pulled up the inclined trough by a weighted cord, running over a pulley at the upper end of the trough to a shaft near the roof, around which it is wound once or twice and kept taut by the weight. To this weight is fastened a second cord, running over a pulley near the roof to the lower end of the trough, which serves to raise the weight, and thus to lower the carriage. In order to prevent the rubber hose from obstructing the upward travel of the carriage and the even flow of the water, it is suspended from the rail by small grooved wheels, and the loops are replaced by 6 iron-pipe return-bends. Thus the suspended hose shows three zigzags, which are close together when the carriage is at the lower end of the trough, and separate as it travels upward, but are held together at the upper ends by strings, which do not allow them to get more than 24 inches apart.

The Cornish automatic feeder is a four-sided truncated pyramid of sheet-iron. It is 24 inches high, and the bases are 18 and 12 inches square. To the smaller base are attached four legs, on which it stands in a sheet-iron box, 16 inches square and 6 inches deep, contracted at one end into a spout. The legs (pieces of angle-iron) firmly connect the hopper and the

box, leaving a distance of ½ inch between them for the ore to pass through. This is charged into the hopper and washed down the spout by a jet of water playing usually between the walls of hopper and box, but occasionally (if especially quick feeding is desired) upon the ore in the hopper.

5. *Jigging.*—The jigs in use for water-sorting are plunger-jigs and movable sieve-jigs. The former are represented by two Collom jigs [10 and 11 and Fig. 3], used for ores ranging from 30- to 5-mesh, the latter by a Richards jig [17] for sizes larger than 5-mesh.

The Collom jigs are two-compartment machines. They are supported by a V-shaped iron frame on each end. The screen-frames are 12½ by 18½ inches. The length of stroke is adjustable to ¾-inch and the number of strokes can be varied by the use of three step-pulleys, 8, 10 and 12 inches in diameter, from 130 to 180 per minute. The ore coming from the feed-trough, the feed-hopper or the spigot of the *Spitzkasten* travels over the jig, while the tailings at the opposite end are collected and unwatered in a sheet-iron box. From this they are drawn at intervals, while the water which overflows goes into the water-tanks [18]. The jigs have no automatic discharge for concentrates; since, for the purposes of instruction and experiment, it is better to stop them every little while and skim off the different layers formed. The manner of working, therefore, is the same as that of large scale one-compartment jigs. The reason for having a two-compartment jig is that "every machine as far as practicable, should have its guard."* Any middle product not remaining on the first sieve will be collected on the second sieve and thus prevented from passing off into the tailings. The Collom jigs here described were put in to replace two three-compartment Harz jigs formerly in use, the screen-frames of which, 6 by 12¾ inches, were much too small to do satisfactory work. The reciprocating motion was derived from an excentric adjustable to 2 inches; and the number of strokes could be varied from 100 to 200 per minute by four step-pulleys, 6, 7½, 9 and 10½ inches in diameter. The jigs had an automatic side-discharge for heads.

The movable sieve-jig serves to illustrate the lectures, to work ore coarser than 5-mesh and to do the water-sorting in graded

* Richards, *Trans.*, xxii., 701.

crushing and jigging. The sieve-frame is 14 inches wide, 22 inches long and 12 inches deep; the ore-bed can reach a depth of 10 inches. The rods of the screen-frame, $\frac{3}{4}$ inch in diameter, are divided into two parts to facilitate taking the machine apart. The two lower or jigging-rods, 48 inches long, are forked at their lower ends and have an eye at the top through which passes a connecting-rod, $\frac{3}{4}$ inch in diameter, suspended from the upper or excentric rods, which are 25 inches long. The excentrics are adjustable to 2 inches, the excentric-shaft is 51 inches long and $1\frac{5}{8}$ inches in diameter. It has a conical pulley with seven steps, its smallest diameter being 6 inches, its largest $8\frac{1}{4}$ inches. The number of strokes per minute ranges from 100 to 200. The counter-shaft is placed 14 inches above the excentric-shaft; and the whole is attached to a strong wooden frame. The water-tank in which the ore is jigged is 33 inches long, 27 inches wide and 22 inches deep. Small boards extending from the sides into the tank serve as guides for the screen frame. The hutch-work is drawn off at the sides; the tank rests on a wooden box and its top is 36 inches from the floor.

6. *Slime-Washing.*—Of the different machines in common use for working slimes [*i.e.*, material not coarser than 30-mesh] only two are represented in the laboratory: a Frue vanner [16] and a convex continuous round table [12]; a greater variety being excluded by the lack of space.

The Frue vanner is of normal size, *i.e.*, it has an inclined rubber surface 4 feet wide and 12 feet long. Either plane or corrugated belts are used. The normal adjustment for full work in the laboratory [inclination of belt $3\frac{1}{2}$ inches in 12 feet, travel of belt 32 inches per minute, and 195 shakes of 1-inch throw per minute] has to be changed, if the pulp flows directly from the light three-stamp battery upon the vanner, as the battery furnishes only about $1\frac{1}{2}$ tons of pulp in twenty-four hours, while the normal rate of the vanner is 5 tons. The simplest way is to change the inclination to $2\frac{1}{2}$ inches in 12 feet and to regulate the flow of water accordingly. If the vanner is to do full work, the pulp from the battery is collected in the settling-tanks and fed at the required rate and with the necessary water by the Hendy feeder of the stamp-battery. In order to permit this, the connecting-rod of the friction-plate is replaced by an

excentric rod, the excentric of which has a 2-inch throw, and is on a small counter-shaft near the ceiling. The counter-shaft is driven from the upper shaft of the laboratory and makes 100 revolutions per minute. The ore which is fed by the carrier-plate is washed by a jet of water into a sheet-iron trough and conducted from behind the mortar into the ore-spreader of the vanner.

The convex continuous round table is 8 feet in diameter and has a slope of $\frac{3}{4}$ inch to the foot. It is of $\frac{1}{8}$-inch sheet-iron, painted with tar, sanded and rubbed smooth, and is supported by an umbrella-frame. It receives its pulp from a fan-shaped distributor, which discharges against one side of a central cone, 14 inches high and 18 inches in diameter, and its wash-water on the opposite side from a horizontal curved pipe with perforations on the inner side. The three products, tailings, middlings and heads, flow into a circular launder. The compartments for heads and middlings are 12 inches wide and hopper-shaped; that for the tailings is 6 inches wide. The heads and middlings are drawn off at intervals into a pail; the water of the heads-compartment overflows into that of the middlings, and the overflow of these into the tailings-launder. The heads are washed off by jets of water; the middlings are sprayed in the usual way. The machine treats from 1 to $1\frac{1}{2}$ tons of ore per day.

There are in the laboratory, of course, the ordinary implements for panning and vanning to check the work done by jigging and slime-washing, and to assist in amalgamating operations.

7. *Electro-Magnetic Separation.*—The magnetic separation of magnetite or of iron-ore rendered magnetic by a preliminary roasting is represented by a small Chase endless-belt machine* placed near the tank [32]. This receives the waste-water from a 6-inch Pelton water-wheel which drives the concentrator. Many interesting data of magnetic separation are recorded in the journal of the laboratory. It may be incidentally remarked that a small Pelton wheel forms a most satisfactory motor for any apparatus that is to be driven independently in a laboratory having water under pressure at its disposal. Of

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* *Trans.*, xxi., 503.

course, a pressure-regulator is necessary to equalize the uneven flow obtaining in a city main.

8. *Dry Concentration.*—There are no arrangements in the laboratory for dry concentration. To make tests that would be in any way satisfactory would require too much space.

9. *Distribution of Power and Water.*—The machinery of the laboratory is driven by a 15 horse-power upright engine [24] having a common slide-valve. Its cylinder is 9 inches in diameter; it has a 9-inch stroke, and is usually run at 200 revolutions per minute. The main-shaft, $1\frac{3}{4}$ inches in diameter, is on the ground-floor and runs the entire length of the milling-room. Its position is approximately indicated by Nos. 1 and 3 in the plan (Fig. 1). It makes 240 revolutions per minute. Near the double ball-grinding mill [36.1] it is connected with the counter-shaft of the same diameter placed near the ceiling. This also runs the entire length of the mill-room along the center-line of the Frue vanner. It makes 200 revolutions per minute. Thus the different machines are set in motion either from the main- or the counter-shaft, the choice depending upon the location and direction of the belts.

The large dynamo [25], an Eddy shunt-wound machine of 50 volts and 50 amperes, is driven at the rate of 2200 revolutions per minute. It has a separate driving-shaft, $1\frac{3}{4}$ inches in diameter, making 550 revolutions per minute. The small dynamo [26], also an Eddy machine of 2 volts and 50 amperes, is connected with a counter-shaft, and makes 1400 revolutions per minute. Electricity has so far been used in the laboratory only for the separation of ores and for the deposition of metals. For electric fusion a differently wound dynamo would have to be added, in order to secure the necessary amperage.

The water required in the laboratory is received from the city main, but is not conducted directly to the different machines, since there would be no regularity in the flow. It runs into the end-compartment of the water-tank [18], from the bottom of which a centrifugal pump, 18 inches in diameter, delivers it into a 2-inch main pipe running along the upper platform, on which are placed the machines Nos. 13, 14, 18, etc. Two-inch tees supply the different machines from the top of the main. By the aid of separate pipes and 3-way cocks the overflow from the jigs can be pumped upon either the

vanner or the round table, the overflow of the vanner upon the table, and the contents of the settling-tanks upon any of the washing-machines or into the sewer.

10. *Auxiliary Apparatus.*—By referring to the plan (Fig. 1) and its legend, the different auxiliary apparatus used in ore-dressing and in metallurgical work can easily be seen. Prominent among these are, for instance, the steam drying-tables [19], on which the products are dried so as to permit comparison of the weights of ore before and after treatment.

The plan does not show the thirty-odd large bins, 4 feet wide, 4 feet deep and 4 feet high, for ores, fluxes, fuels and intermediary products. They are accessible from the furnace-room by two doors, and from the milling-room by one door.

B.—Sampling- and Assaying-Apparatus.

Ore-sampling is generally done in the laboratory by hand. If it is desirable to do mechanical sampling, only intermittent machines—those which take the whole of a stream of ore at stated intervals—are allowable. The small-size machines of Bridgman and Constant do good work. Ores are crushed in rock-breakers and rolls and pulverized in the Hendrie and Bolthoff sample-grinder [5] or on bucking-plates [20]. Samples for analytical purposes are ground fine in four Morrel agate-mortars [24.1]. The ores are all sampled by hand on the iron sampling-floor [6] or on the sampling-table [21]. Liquid pulp, fed upon or coming from washing-machines, is passed through specially-constructed automatic samplers (see *e.g.*, Fig. 2). Samples from alloys are taken by chipping, punching, sawing and boring [35]. In laboratory instruction too little stress is apt to be laid on the sampling of ores and metallurgical products. It is a most important and necessary part of the work, the whole of which is really invalidated if the sampling is inaccurate.

Assaying, in its broadest meaning, includes the quick quantitative determination of any element or compound met with in metallurgical work, embracing not only fire-assays but also what is known as analytical work on solids, liquids and gases. In the Institute metallurgical laboratory assaying is restricted to fire-work (except as regards the parting of doré silver buttons or chlorination assays). All analytical work is done in the

chemical laboratories. The assay-laboratory has two divisions: the assay-room proper [37], and the balance-room [44]. The assay-room has eight pulp-balances [39], weighing accurately to 1 milligramme with a load of 60 grammes, and six flux-balances, accurate to 0.1 gramme with a load of 600 grammes. They are distributed among the students' desks [38], of which there are fifty. There are twelve crucible-furnaces [41]; nine muffle-furnaces [40], three of which have lately been erected in "the space to grow" [63]; and, lastly, an iron table [43] for hot crucibles, etc. Under the table is a shelf for crucible and scorifier moulds, and beneath this are small bins for fuels. Along the side of the table are four posts, with anvils for breaking crucibles, hammering buttons, etc. The crucible-furnaces are 27 inches high and 12 by 12 inches in the clear. They are inclosed in wrought-iron plates, and thus firmly held together. The top of each furnace is horizontal, and is covered by a fire-clay tile, around which is shrunk an iron band, with two hooks riveted to it. The cover is suspended from a wire cord passing over a pulley attached to the ceiling, a counterweight being at the other end.

The muffle-furnaces are of different kinds and sizes. Five are Judson coke-furnaces, two with muffles, 4 by 7 inches, closed at one end, and three with muffles 8 by 16 inches, open at both ends; also, three coke-furnaces, with sheet-iron housing and fire-brick lining, having muffles 7 by 12 inches, closed at one end; and, lastly, one two-muffle furnace for bituminous coal, with muffles, 6 by 13 inches, open at both ends. Oil- and gas-furnaces are not used. The draft for all the furnaces is furnished by one main chimney [42], 2 by 3 feet, and about 80 feet high.

The balance-room contains one analytical balance and nine button-balances [45]. The principal aim has been to have the leading makers, such as Ainsworth, Becker, Oertling, Troemner and others, represented. The balances are accurate to 0.01 milligramme, with a maximum load of 0.5 gramme.

C.—Metallurgical Apparatus.

While the various operations of the concentration of ores and fuels can be carried on in a school- or general experimental laboratory so as to give practical results, the case is likely to be

somewhat altered when it comes to metallurgical processes. If we take, *e.g.*, a leading process—that of smelting in the blast-furnace, we cannot reduce the operations to a laboratory-scale, and obtain results which will serve as a guide for practical work. Nevertheless, smelting in the blast-furnace ought to be a part of the laboratory work, on account of its educational value. If a student receives for treatment a batch of ore, examines it mineralogically and chemically, makes the necessary analytical determinations of his fluxes and fuel, calculates his charge, smelts it and sums up his results by weighing, assaying and analyzing the products, he learns more about smelting than any amount of lecturing or cursory visiting of works can ever teach him. Only by taking hold himself and carrying a process through to the end, can he learn how to think metallurgically, and thus become really qualified to listen intelligently to what is taught in the class-room.

There are, however, many metallurgical processes—such as roasting, amalgamating, leaching, electro-deposition and other operations—which can be performed in the laboratory on a small scale with trustworthy economic results. In fact, the engineer is guided, in the planning of amalgamating-, and leaching-mills, by the results obtained in such laboratory-experiments. This class of work should therefore have a prominent place in the laboratory. From what has been said, it will be evident that most operations relating to the metallurgy of iron and steel must be excluded. Attempts have been made to imitate large-scale iron and steel-work in the laboratory. For instance, the Sheffield Technical School, in England, has a small open-hearth steel-furnace; the Polytechnic School of Aix-la-Chapelle, Germany, has a small puddling-furnace; but the writer, though not acquainted with the results obtained, is much inclined to doubt whether they will be found to justify the large outlay of time and labor involved. We must always keep in mind that it is not the province of an engineering school to perfect the student in any one branch of his profession, so much as to ground him in the fundamental principles upon which he is, later, to build for himself in detail.

In the laboratory of the Institute the processes chosen for instruction are those involved in the treatment of lead-, copper-, gold- and silver-ores and the ores of some of the minor metals.

although it should be added that crucible-work and other small-scale heat-treatment of iron and steel, especially with regard to their physical properties, are not excluded.

The furnace-room [48] contains apparatus enough of various kinds to carry on all the necessary operations, so arranged as to occupy as little space as possible. This forces a crowding of the furnaces; but as the work can be so laid out that adjoining furnaces need not be used at the same time, less inconvenience results than might be at first supposed. The necessary draft is furnished by a stack [84] 2 by 3 feet and about 80 feet high. A horizontal main flue, 3 by 3 feet, running along three sides of the room—sometimes near the ground, sometimes near the ceiling, according to the height of the furnaces—collects the gases. Each furnace, however, can be shut off from it by a damper in its branch-flue. Too much stress can hardly be laid upon the necessity of securing a strong draft. The main and branch-flues should be large, and the stack of ample section and sufficient height, so that it shall be possible to run each of the furnaces alone or any number or all of them together. With a well-fitting damper, it is an easy matter to cut off too much draft; if there is too little, the result is fatal.

1. *Roasting.*—For this purpose there are three reverberatory furnaces and one stall.

The large hand-reverberatory [56] covers 8 feet 2 inches by 5 feet 7 inches, and is 4 feet 8 inches high. Its hearth is 4 feet 2 inches long and 3 feet wide, and lies 9½ inches below the top of the fire-bridge, which is 9 inches wide. The height of the 9-inch side-wall is 11 inches to the spring of the arch, the height of the arch 5 inches. The furnace has one working-door, 14 by 9 inches in size, and 2 feet 10 inches from the ground. The gases pass off through three openings, 9 by 9 inches, in the roof, into a branch flue running across the furnace and ending in the main flue. The fire-place, 2 feet 3 inches by 1 foot 9 inches, lies 16 inches below the top of the bridge, which is 8 inches below the roof. It has a door 12 by 9 inches in size, and 2 feet 6 inches from the ground. The furnace treats charges of about 250 pounds of pyritic ore.

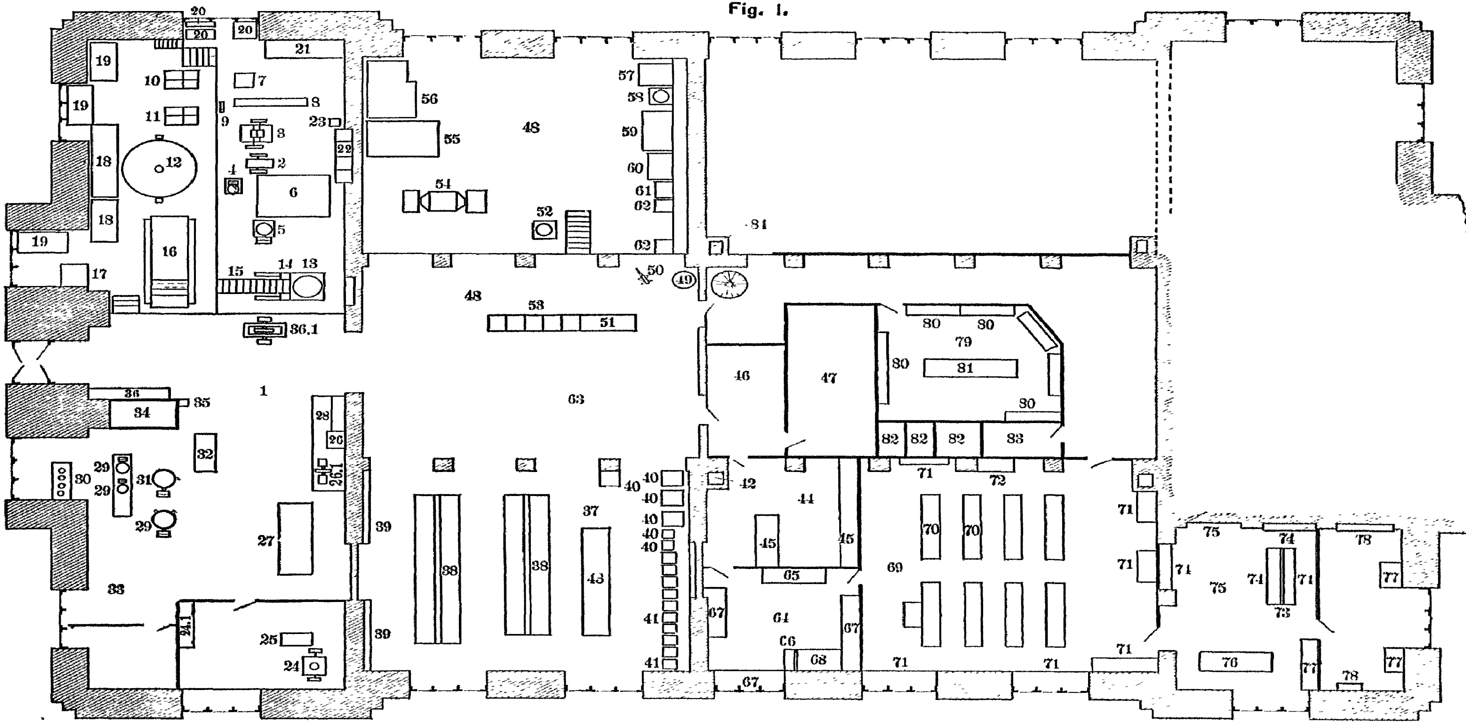
The outside dimensions of the small hand-reverberatory [60] are: Length, 8 feet; width, 2 feet 8 inches; height, 5 feet. The hearth is 2 feet square and 6½ inches below the top of the

bridge, which is 3 inches wide. The height of the $4\frac{1}{2}$-inch side-wall is 8 inches to the spring of the arch, and that of the arch is $5\frac{1}{2}$ inches. The working-door is 9 by 6 inches, and 2 feet 10 inches from the floor; the flue running over the furnace is 5 inches square. The fire-place, 1 by 2 feet, is 10 inches below the top of the bridge, which is 7 inches below the roof; and its door, 9 by 6 inches, is 2 feet 6 inches above the floor. The furnace works small charges of, say, 25 pounds of pyritic ore.

The drawback of roasting in such small reverberatories is that the charge is liable to become too much cooled near the working-door. If there had been more room, both roasting-furnaces would have been constructed, like the reverberatory smelting-furnace, with the working-door at the end and the flue just above it; the air necessary for roasting being admitted through the hollow bridge. It might also be an improvement to have the hearth built in an iron pan, and so arranged as to permit its being removed, cleaned, and examined after an operation; although this is not so necessary in roasting as in smelting.

The third reverberatory roasting-furnace, the Brückner cylinder [54 and Fig. 4], gives opportunity to study the behavior of an ore on a revolving hearth. The outside dimensions are: Length, 6 feet, and diameter 2 feet $8\frac{1}{2}$ inches. The cylinder is of $\frac{3}{16}$-inch boiler-iron, and has a $2\frac{1}{2}$-inch fire-brick lining. The throat is 12 inches, and the charging-hole 8 inches in diameter. The cylinder, the axis of which is 3 feet 5 inches above the ground, revolves on two iron friction-rings (35 inches in diameter) which rest on four 12-inch carrying-rollers. One of the carrying-roller shafts ($2\frac{7}{16}$ inches in diameter) is rotated by a worm-gear (62 teeth of 1-inch pitch) at the rate of 20 revolutions per hour. The fire-box is detached and rests on castor-wheels. By moving the box backwards or sidewise, the amount of air admitted can be increased. An additional improvement would be to make the throat of the fire-box muffle-shaped, leaving that of the furnace circular. In order to have complete control over the flame, the grate (18 by 24 inches) is laid 20 inches below the bridge. The carbonic oxide gas generated is burned by warmed air entering the furnace just above the bridge, after having been forced through five flues in the side-wall and roof of the fire-box. The ash-pit, 8 inches deep, is closed and con-

Fig. 1.

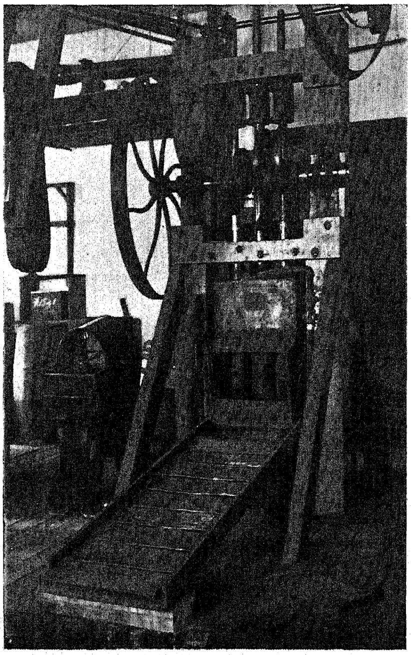


PLAN OF JOHN CUMMINGS LABORATORY OF MINING ENGINEERING AND METALLURGY.

Am. Bank Note Co. N.Y.

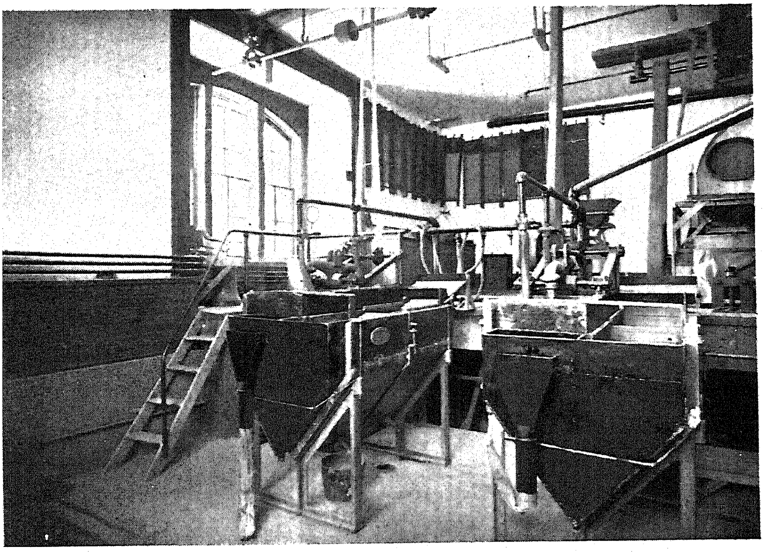
0 20 40 60 80 100 FT.

Fig. 2.



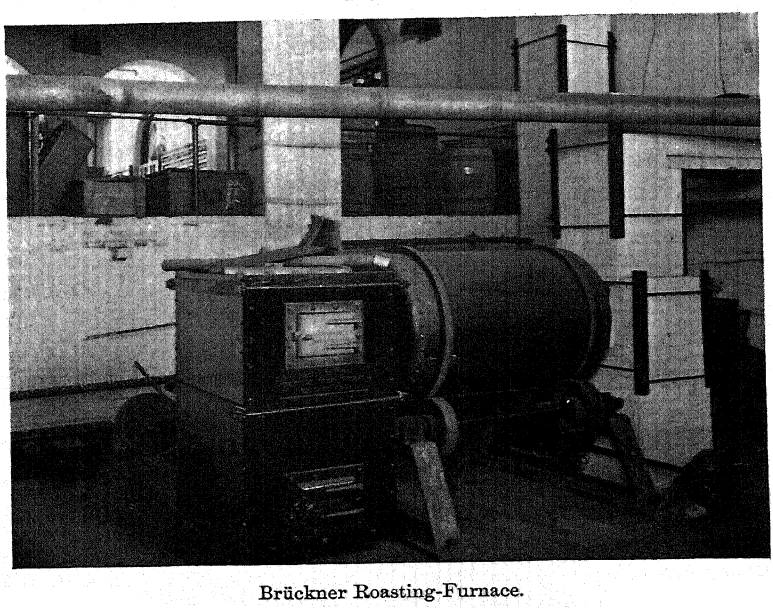
Stamp-Battery and Amalgamated Plates.

Fig. 3.



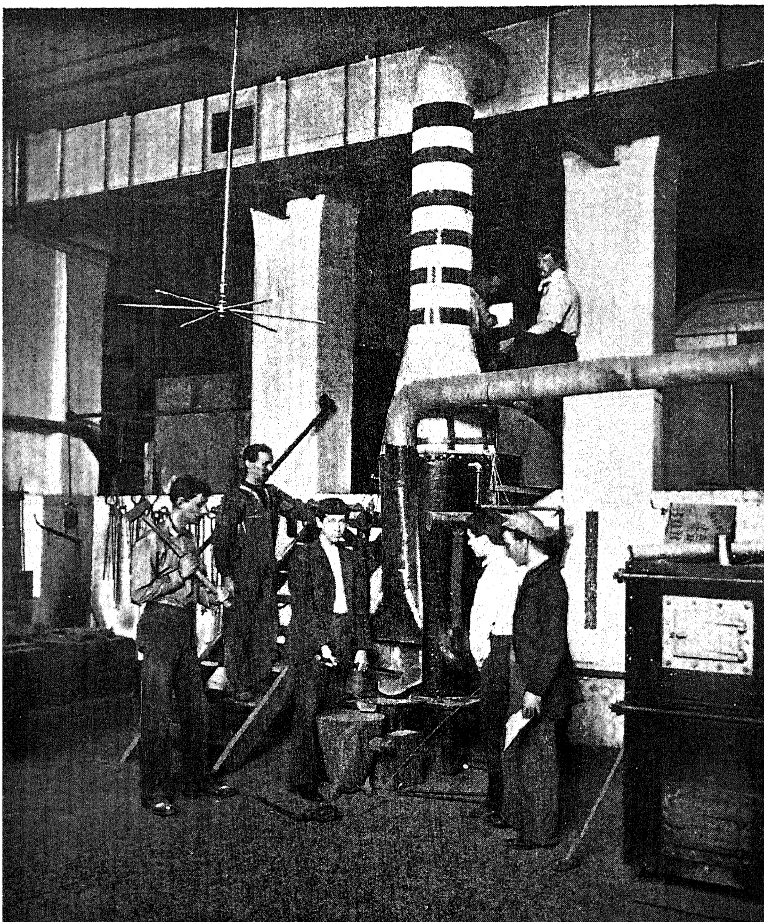
A Pair of Two-Sieve Collom Jigs.

Fig. 4.



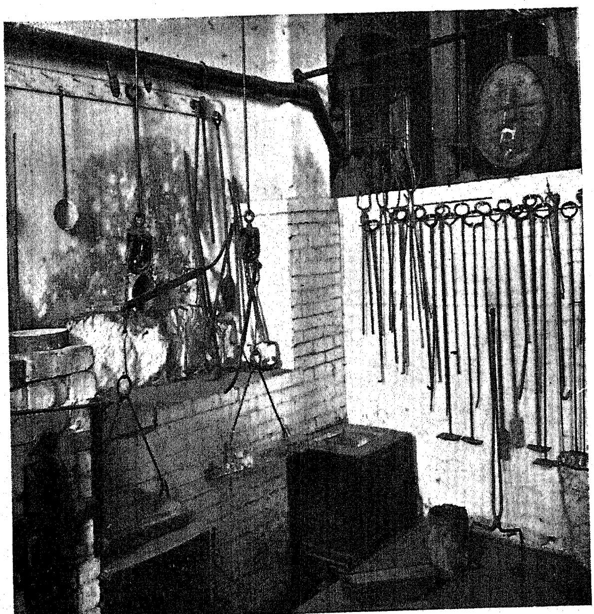
Brückner Roasting-Furnace.

Fig. 5.



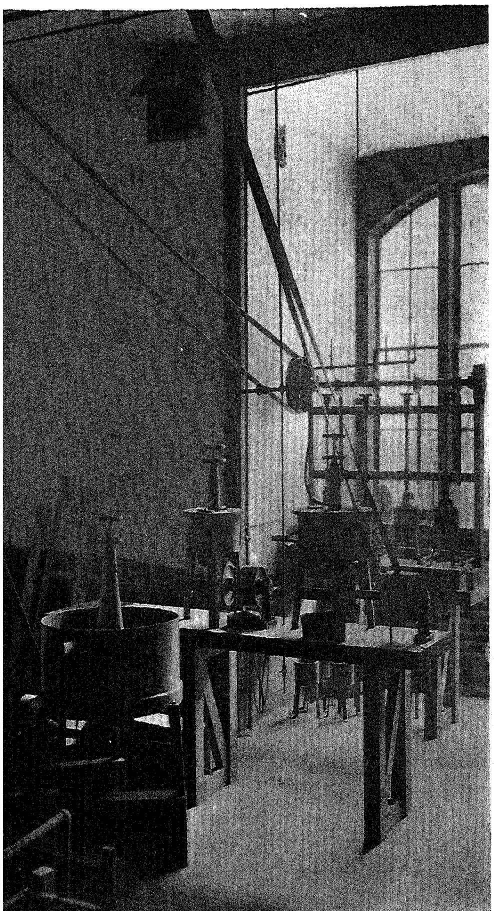
Water-Jacket Furnace for Smelting Lead and Copper Ores.

Fig. 6.

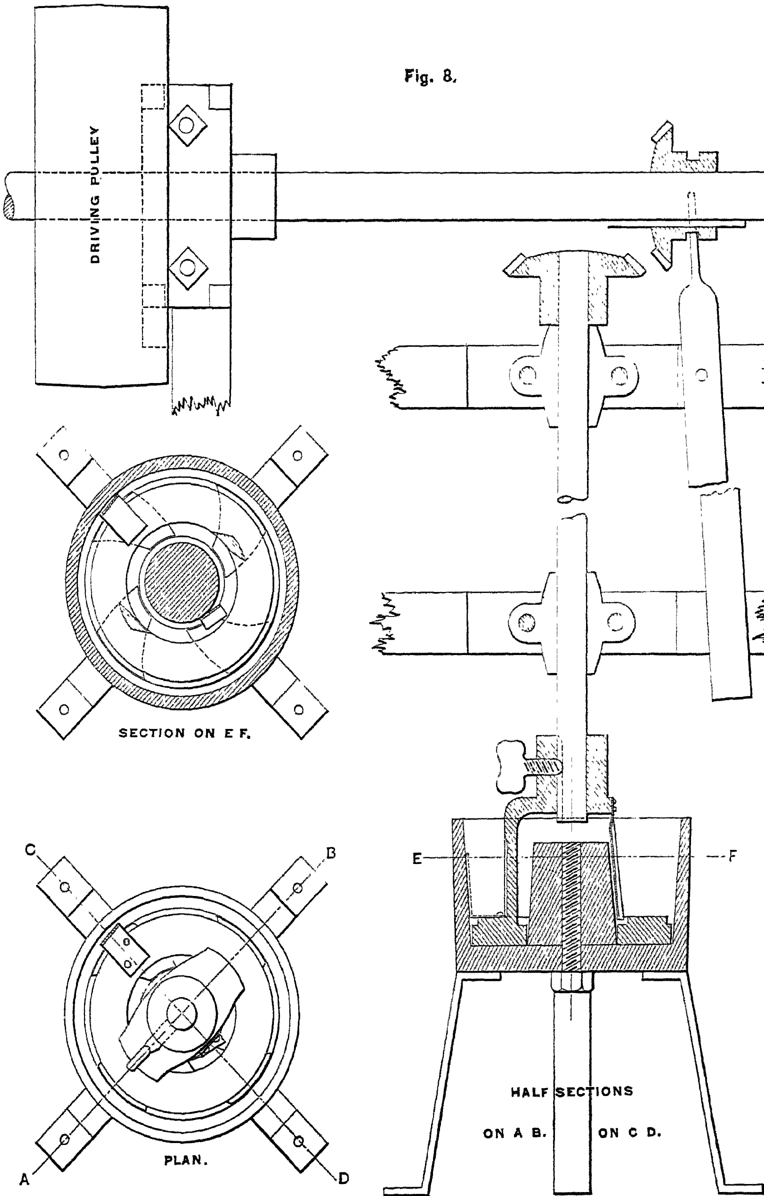


Furnaces for Pot-Melting, with Travelling Lift for Covers.

FIG. 7.

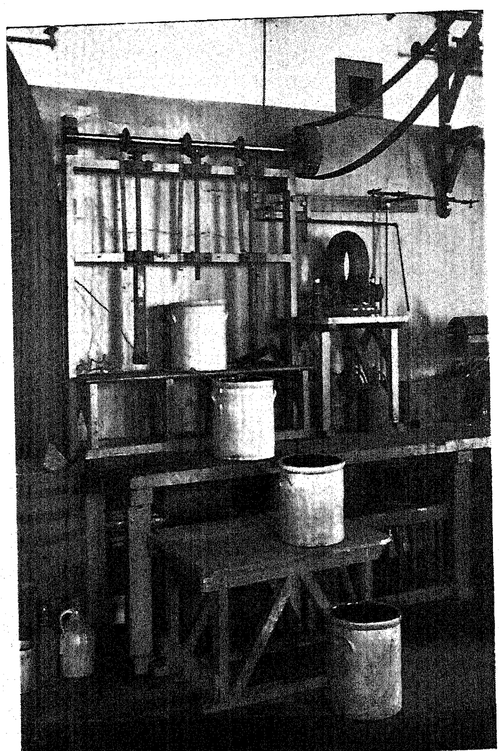


Laboratory Amalgamating-Pans.



RICHARDS LABORATORY AMALGAMATING-PAN.
SCALE: 2 inches = 1 foot

Fig. 9.



Leaching-Tubs Arranged for Mechanical Stirring.

nected with a blast-pipe. This furnace treats charges of about 200 pounds.

The stall [57], which completes the roasting apparatus, is commonly used for treating coarse copper-bearing pyrites previous to smelting in the blast-furnace. It is 3 feet 3 inches deep, 2 feet 3 inches wide and 3 feet 7 inches high to the spring of the arch. The arch is 6 inches high. The walls are 4 inches thick and well anchored. The ore is roasted on a temporary grate of wrought-iron bars. The front is bricked up half-way, the upper half being closed by an iron plate with peep-hole. The charge varies from 1500 to 2000 pounds, and a roast lasts from two to three days. The results in desulphurization are very similar to those in large stalls. The management of the stall affords a splendid lesson in the regulation of draft.

2. *Smelting.*—Smelting is carried on in the blast-furnace, the reverberatory furnace and the crucible-furnace.

The blast-furnace [52 and Fig. 5] has had to undergo several changes before it reached the present satisfactory form. The first furnace, 18 by 15 inches at the tuyere level, was built of brick. It had one tuyere at the back, run with a "nose," the ore being charged towards the back and the fuel towards the front. It would last one day, or perhaps two days, and then had to be relined. The next furnace, 18 by 16 inches, with three ordinary tuyeres, and charged in horizontal layers, burned out in less than a day. When provided, however, with one water-cooled tuyere at the back, projecting 8 inches, it was run successfully, and had to be relined only once a year. With this furnace ores were smelted for about six years, until, in 1884, the present one replaced it. This is a water-jacket furnace, resembling the circular copper-smelter in common use to-day. The height of the furnace, 6 feet 6½ inches, is divided as follows: height of four hollow cast-iron columns, 17½ inches; thickness of annular collar, 1 inch; distance to tuyeres, 1 foot; diameter of tuyeres, 2 inches, and height to feed-door, 3 feet 10 inches. The diameter at the bed-plate is 1 foot 5 inches; at the tuyeres, 1 foot 6 inches; at the throat, 1 foot 11 inches. The furnace has a conical hood 2 feet 9 inches high and 25 and 11 inches in diameter, which ends in a vertical flue leading into the main flue. The feed-door is 13 inches high, 14 and 9¾

inches wide. The water-jacket is of $\frac{3}{8}$-inch boiler-iron and has a 3-inch water-space. The feed-water is supplied from the city main through a $\frac{3}{4}$-inch pipe near the top, the overflow-pipe being tapped into the upper flange. There are four tuyere-holes, lined with solid bored blocks of bronze. The tuyere-pipes are of wrought-iron steam-pipe; the horizontal arm has at one end a conical turned bronze nozzle, at the other a T, the vertical leg of which is connected by a pipe with the tuyere-bag, and the horizontal leg, reduced in diameter by a bushing, is closed with a cap having a glass-covered peep-hole. The bustle-pipe is 4 inches in diameter. The bottom of the furnace is closed by a wrought-iron plate clamped to the collar of the four columns. The crucible is lined with brasque tamped in solid from above to the level of the tuyeres, and then cut out from below into the desired shape, the lining reaching up to the tuyeres.

In tapping the melted masses from the furnace different methods were tried before the present one was adopted. With an internal crucible and separate metal- and slag-taps the metal easily became cool; with an external crucible and continuous flow it cooled even more quickly. The present practice is to tap the melted masses into a small cast-iron overflow-pot, having the form of an inverted pyramid, 6 inches deep, $12\frac{1}{2}$ inches square at the top and $2\frac{1}{2}$ inches square at the bottom. This retains the metal, matte and foul slag, and is removed after every tapping by means of iron hooks inserted through rings on either side. The clean slag overflows into an ordinary conical slag-pot, 14 inches in diameter and $16\frac{1}{2}$ inches deep. A detached carriage serves to take away the full pots and return the empty ones. A Devereux slag-pot may in the future replace the arrangement now in use. The fumes from tap-hole and slag-pot are drawn off by a hood connected with a small fan. The furnace has a daily smelting-capacity of about 6 tons of charge, not counting the fuel. It is not run, however, for 24 hours at a time. The furnace, warmed during the preceding day and night, is usually blown-in at 8 A.M. and blown down again about 4 P.M. This period is sufficient to give the student all the instruction that he can get from carrying on a smelting operation on such a small scale. Longer runs would mean greater physical exertion without corresponding

benefit. When a run is completed, all the products are carefully separated and, if necessary, the matte adhering to foul slag or metal is separated by an additional crucible-fusion, and thus a complete account of stock is taken. With the present arrangements the loss of metal in flue-dust has to be arrived at indirectly by difference. It is proposed, however, to save the flue-dust, either by cooling or filtering or by wet condensation, and thus to obtain direct figures.

Three reverberatory smelting-furnaces were once considered necessary to fill the wants of the laboratory for agglomerating lead and copper-ores, smelting lead-ores, cupelling base bullion, bringing forward matte and refining copper. Two furnaces are sufficient. The English cupelling-furnace [59] serves for the last three operations; while the other two, formerly carried on in a reverberatory furnace (replaced to-day by the Brückner cylinder) will be taken up again when the copper-refining furnace [55] has been rebuilt as a reverberatory furnace with movable hearth inclined from bridge to flue. The cupelling-furnace is of the ordinary pattern. The test is 18 by 24 inches, and is wedged fast against the test-ring; the fire-place, 18 by 24 inches, is run with the under-wind; the grate is laid low, 20 inches below the top of the bridge, which is 9 inches wide and 15 inches below the roof. In order to burn the carbonic oxide gas formed there is a special tuyere in the side of the furnace just above the level of the bridge. In addition to the tuyere at the back of the hearth, there is a second one in the roof connected with a U-shaped pipe passing through the flue. Hot blast comes into play when a quick raising of the temperature is desired. The different kinds of reverberatory work so far practiced in this furnace, such as liquating drosses on an iron plate, softening and cupelling base bullion on a hearth of limestone and clay, concentrating matte and refining copper on a hearth of a mixture of raw and burnt fire-clay or closely-fitted refractory tiles have been so satisfactory that the idea of a fixed hearth for laboratory-purposes has been entirely given up. In the furnace 150 pounds of base bullion, assaying about 150 ounces of silver per ton, are cupelled in 6 hours, or 200 pounds of black copper are brought through the different stages to tough-pitch copper in 7 hours.

The plan, Fig. 1, shows a small cupelling-furnace [61], which

is used sometimes to refine impure silver from the English cupelling-furnace in quantities larger than can be satisfactorily treated in one of the muffle-furnaces. It has a small fire-place, 8 by 14 inches, and 15 inches deep, the flame rising from which strikes the fire-clay tile forming the roof, and is deflected so as to strike the silver (placed in an oval cupel-test, 8 by 14 inches, and 2 inches deep, filled with bone-ash).

Crucible-work is of considerable importance in a metallurgical laboratory, as it is not only adapted for independent experiments, but serves to bring into suitable form the different mixed products obtained in the processes carried out on a larger scale in the laboratory. Small crucibles are commonly heated in the assay-furnaces; for larger charges there are two pot-furnaces [62, and Fig. 6], worked with under-wind. They are 14 inches square and 23 inches deep; the blast is introduced through the ash-pit door, and the ash-pit is 9 inches deep. A furnace holds conveniently a No. 35 graphite crucible.

3. *Distillation and Sublimation.*—Both these operations are of subordinate importance in laboratory-work. Distillation of mercury is carried on in half-pint and one-pint bulb-retorts, which are heated over four-tube Bunsen burners. The delivery-pipe is cooled by suspending from it an iron trough filled with cotton waste, which is kept wet. Reduction of zinc oxide or sublimation of arsenic, realgar and sulphur are rare operations, and no special apparatus is assigned for this purpose.

4. *Crystallization.*—The principal process coming under this head is the Pattinson process, for which a cast-iron kettle [58] is used, 21 inches in diameter and 14 inches deep, covered with a hood and heated by a fire-place 21 inches square. This kettle is rather small for the Pattinson process; it is the one in common use for desilverizing argentiferous lead by the Parkes process, and for melting and liquating, in general, readily fusible metals and alloys.

5. *Amalgamation.*—The process of amalgamation is especially well adapted for laboratory-work, since small-scale experiments give results directly applicable to large-scale work. The different appliances for treating gold- and silver-ores in this way are therefore well represented. There are a stamp-battery, a ball-mill, two revolving barrels and a number of pans of different sizes.

The stamp-battery, as a pulverizer, has already been described under the head of fine-crushing. In using it for the amalgamation of gold-ores, the arrangement and management of the copper plates (see Fig. 2) differs from that of large-scale work in having nine small plates, 24 by 11 inches, and $\frac{1}{16}$ inch thick, laid cross-wise over the apron-table, one overlapping the other,* instead of a single large sheet of copper, and also in not having inside plates. By having several outside plates, and cleaning them up separately, it can be seen how the gold saved decreases with the distance from the mortar-discharge, and the required length of plate can thus be determined. In order to prevent absorption of gold by the outside plate, it is coated with silver-amalgam. On an inside plate this would be scoured off and gold would be absorbed by the copper, thus vitiating the test; hence, inside plates are not recommended.

The ball-mill [36.1] is used for grinding and amalgamating small lots of gold-ore and for cleaning up the battery-residues. The plan shows a circular cast-iron [$\frac{3}{4}$-inch] plate, 22 inches in diameter, on each end of a horizontal shaft, 2 inches in diameter and 27 inches long, in the center of which is the driving pulley, 20 inches in diameter. To each plate is bolted a flanged cylindrical box (7 inches deep, 17 inches in diameter and $1\frac{1}{2}$ inches thick), having a 4-inch charging-hole opposite the shaft, to be closed by a wooden bung, and a $1\frac{1}{4}$-inch discharge-opening, to be closed by a screw-plug. From thirty to forty $1\frac{1}{2}$-inch steel balls do the grinding. The mill makes 48 revolutions per minute, and works two charges of 15 pounds of ore in about ten hours.

The revolving barrel [26.1] serves for amalgamating without grinding, as well as for leaching. Its general arrangement is similar to that of the ball mill. To each end of the horizontal shaft, $1\frac{1}{2}$ inches in diameter and driven by a 20-inch pulley, is attached a wooden cylinder, 7 inches in diameter and 11 inches long, made of $\frac{7}{8}$-inch staves which receives a 2-quart glass-stoppered fruit-jar, made tight with a rubber washer and screw-clamp. The jar is packed with felt into the wooden frame. The shaft makes from 20 to 25 revolutions per minute. Small lots of ore, of 1000 grammes, more or less, are worked in about eight hours.

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* Richards, *Trans.*, viii., 362; *Technology Quarterly*, iii., 45; Editorial, *Engineering and Mining Journal*, April 12, 1890, vol. xlix., p. 418.

There are ten amalgamating-pans [29 and 30, Figs. 7 and 8]. Three of these are accurate copies, in reduced size, of those used in practical work. They are 30, 18 and 12 inches in diameter, have sides 12, 8 and 6 inches deep and discharge into a 30-inch settler, 12 inches deep, making 15 revolutions per minute. They treat charges of 250, 30 and 20 pounds, respectively, in from five to eight hours. The other seven pans [Fig. 8] especially constructed for laboratory-experiments, are only 7 inches in diameter. Three of these are of copper; the others of iron. The pan has a solid central core and no dies; the muller and shoes are cast in one; the pulp is prevented from settling on the core and sides by adjustable scrapers. The muller can be raised or lowered on the driving-shaft, which is driven from above and easily thrown in and out of gear. The pans are heated by Bunsen burners. The muller makes 90 revolutions per minute and the pan works charges in three or more hours. The reason for choosing such small-sized pans is that in one day's work, two students will finish without outside help a set of experiments. They start, for example, in the morning, four pans with the same ore, treat it in four different ways and finish the cleaning-up in the afternoon. A larger pan or a pan of a more complicated construction will not permit this. In cleaning up, a large-sized *Spitzlutte*, 3¾ inches in diameter and 13 inches high, with a ¾-inch water inlet-pipe is commonly used, as it does quick and effective work.

6. *Lixiviation.*—The leaching of ores and intermediary products can be done in the laboratory in stationary vats by percolation, or by mechanical stirring, or in revolving barrels. For leaching by percolation there are two forty-gallon vats (not shown in the plan, Fig. 1) of wood lined with lead. These will be replaced with sheet-iron vats poured with melted roofing-pitch. For leaching in stationary vats with mechanical stirring there are three sets of 8-gallon vessels [28 and Fig. 9] of glazed earthenware, 12 inches in diameter and 14 inches deep. The wooden stirrers, with their iron driving-shafts, make 75 revolutions per minute. For leaching in a revolving barrel the same apparatus is used as for amalgamation. Gold-, silver- and copper-ores are commonly, and zinc- and nickel-ores occasionally, treated by wet processes in the laboratory.

7. *Electro-Metallurgical Work.*—Electricity has so far been

used only for the refining of silver- and gold-bearing copper. The large depositing-table [27] holds the electrolytic baths. They are of wood-pulp, poured with melted roofing-pitch, of glass or of earthenware, as the case may be. No definite sizes have been, so far, adopted, but electrodes are usually made 7 by 10 inches. The current is furnished by the dynamos already referred to; thermo-piles and storage batteries are not in use.

D.—Conclusion.

It is somewhat difficult to estimate the cost of the laboratory-apparatus, because one thing has been put in after another, and alterations have been frequently made. It could probably be duplicated for about $15,000. The annual cost of running the laboratory, excluding wages, fuel and power, is $1200.

That it is conducted in connection with class-room work, and not independently, need hardly be mentioned. With the school-courses of the fourth year the students are thoroughly trained in the laboratory, their work there supplementing and illustrating the lectures. The last term is largely devoted to the working up of theses, which are always founded on laboratory-experiment. While the student does not handle every apparatus, he sees most of them in operation. Every Saturday each student makes, before the assembled class, an oral report of his laboratory-work during the past week, and its continuation for the coming one is discussed and laid out. The whole class thus gets the benefit of the work of each individual member. The time devoted to laboratory-work is 325 hours, and to class-room work, including preparation, during the same year, 225 hours. The most satisfactory arrangement would be to have during the entire year two days a week for laboratory-work. One of these should be uninterrupted for making a complete experiment, the other might be divided into two half-days.

Folds and Faults in Pennsylvania Anthracite-Beds.

BY BENJAMIN SMITH LYMAN, PHILADELPHIA, PA.

(Atlanta Meeting, October, 1895.)

It has seemed that it might be a highly useful contribution to the study of structural geology to assemble, in as compact a form and on as large a scale as practicable, a great number of

cross-sections of actual workings in the Pennsylvania anthracite-beds. Accordingly the accompanying thirty-three page-plates, containing 177 sections, have been prepared from the numerous very valuable cross-section sheets of the State Geological Survey, besides a key-map, to show where the sections were made.

The sections are all in northwesterly and southeasterly directions, and are looked at from the southwest. They are drawn with equal horizontal and vertical scales, and are consequently not misleading, as distorted ones would almost necessarily be. They have been reduced by photography from the original scale of 400 feet to the inch to one of 500 feet to the inch, except a couple of them on much larger scales. Initial letters and numbers on each section refer to the section and sheet of the original Atlases of the Northern, Eastern Middle, Western Middle, and Southern anthracite-fields. The geographical position of each section is also indicated by its distance and direction from the nearest large town. The portions of a bed worked out on the line of the sections are represented by two light lines, indicating the top and bottom of the bed; but where the space is filled with black, the position of the bed is well ascertained, and where there is less certainty, the bed is represented by a single light line.

An inspection and tabulation of the general forms of the principal and subordinate basins and saddles of the cross-sections, show that the law that the northwesterly dips are steeper than the southeasterly ones, is not by any means so universal as H. D. Rogers seems to say in his able discussion of the laws of geological structure in the State Geological Report of 1858. On page 889 of vol ii., he says:

"Almost invariably, those [the flexures] of a simply undulated tract exhibit their steeper slopes directed all to one quarter."

Again, page 894, he says;

"There exist among the undulations of the strata in Pennsylvania a few—they are very few—exceptions to the almost universal law of a superior degree of abruptness of incurvation upon the northwest slopes of the anticlinal waves. These abnormal instances of relative dip belong almost invariably to the secondary class of flexures, which I have never regarded as true waves pervading the earth's crust, but as comparatively superficial foldings occasioned by the joint agency of pulsation and lateral crumpling. There are a few examples of unusual steepness of the southeast dips in the primary class of flexures; but nearly every

one of these exceptions applies to only a local portion of the wave, and will be found connected either with a fault in the strata, or with an oblique interference of the end of an anticlinal of another group. I think there does not exist within the whole wide undulated zone of the State, or of the Appalachian chain generally, a wave or group of waves of the first order, which is abnormal as respects the direction of the flatter and steeper slopes, except where we can directly refer it to the influence of some prodigious crust-dislocation. . . . . Of the lesser class of flexures of a reversed profile, we may instance many in the anthracite coal-fields, particularly in the Shamokin basin."

The cross-sections in our plates are obviously not selected, consciously or unconsciously, with any reference to the form of the basin, and might, so far as that is concerned, be considered as made quite at random, and as likely, therefore, to indicate in an unprejudiced way what may be the general laws of the forms of the basins and saddles. It is true that the number of the undulations is not large enough in some of the subdivisions of the anthracite-region to be considered a perfectly precise indication of the relative number of the different forms; but the result for the 177 sections in the whole region, with 500 saddles and basins, large and small—nearly half of them (219) large—may probably be accepted as giving some useful indications. Taking them all together, it appears that nearly half of them have about equal dips on the two sides, and about as large a number have steeper northwesterly dips, and one-third as many have steeper southeasterly dips; or more precisely, three-sevenths of the whole number have the two dips about equal; three-sevenths steeper northwesterly dips, and one-seventh steeper southeasterly dips; or 43 per cent., $42\frac{1}{2}$ per cent. and $14\frac{1}{2}$ per cent. respectively.

In order to ascertain what the corresponding proportions might be in the four different fields of the region, and in the smaller folds as distinguished from the larger ones, so far as the small number of sections could give indications, tables were made showing the number and percentages of the larger or main basins, and of the main saddles, and of the smaller or subordinate ones in each field, that have about equal dips on the two sides, or steeper northwesterly dips or steeper southeasterly ones. The division into main and subordinate folds may not in every instance be accurate, but probably is sufficiently exact for the purpose, or to correspond with the exactness of the comparison in other respects. In some cases, adjacent

basins and saddles have a dip in common; but that also would probably not influence the main results. As might be expected, in each group, main or subordinate, the percentages of the different forms in the basins were about equal to those in the saddles; and very closely equal in the large number of cases supplied by the whole region, less closely, it is true, for the scantier cases of the different fields. It is only needful, then, to give here a single table of the percentages in the different fields, and in the two classes of main and subordinate folds (whether basins or saddles), as follows:

Percentages of Equal and Steeper Dips.

	Main Folds.			Subordinate Folds.		
	Equal Dips.	North Steep.	South Steep.	Equal Dips.	North Steep.	South Steep.
Northern Anthracite-field, .	58½	27¼	14¼	40½	38½	21
Eastern Middle " .	33½	53	13½	37¾	51½	10¾
Western Middle " .	45	43½	11½	71	20½	8½
Southern Middle " .	21	60	19	50	37½	12½
All, ...	37½	48	14½	48	38	14

The table shows that in the whole anthracite region only three-eighths of the larger folds are symmetrical, while about half of them have steeper northerly dips, and about one-seventh steeper southerly ones; but that about half of the smaller folds are symmetrical, three-eighths have steeper northerly dips, and one-seventh have steeper southerly ones. In the different fields, however, the results would seem plainly to be different. In the Northern anthracite-field, decidedly more than half of the larger folds are symmetrical, only about half as many have steeper northerly dips, and about a quarter as many have steeper southerly dips; while of the subordinate folds about two-fifths are symmetrical, a nearly equal number have steeper northerly dips, and half as many have steeper southerly dips. In the Eastern Middle field one-third of the larger folds are symmetrical, over half have steeper northerly dips, and about one-seventh have steeper southerly dips; but of the subordinate folds, three-eighths are symmetrical, one-half have steeper northerly dips, and one-ninth have steeper southerly ones. In the Western Middle field rather less than half of the larger folds are symmetrical, about an equal number have steeper northerly dips, and about one-ninth have steeper southerly ones; but of the sub-

ordinate folds nearly three-quarters are symmetrical, only one-fifth have steeper northerly dips, and one-twelfth have steeper southerly ones. Finally, in the Southern field, only about one-fifth of the larger folds are symmetrical, nearly two-thirds have steeper northerly dips, and about one-fifth have steeper southerly ones; but of the subordinate folds one-quarter are symmetrical, three-eighths have steeper northerly dips, and one-eighth steeper southerly ones.

Some at least of the variations in the different groups seem to be quite too decided to be due merely to an accidental deficiency in the number of cases. There seems to be a strong resemblance between the Eastern Middle field and the Southern one in the figures for the larger folds, though not for the smaller ones; and it is only in these fields that the steeper northerly dips of the larger folds are much more numerous than the symmetrical folds. The Eastern Middle field is the only one where the smaller folds have more northerly steep dips than symmetrical ones. The Northern field has a remarkable number of symmetrical larger folds; and the Western Middle field a very remarkable number of symmetrical smaller folds, and very few with steeper northerly dips. In each group the steeper southerly dips are about a quarter or a third of the steeper northerly ones, except rather more in the smaller folds of the Northern and Western Middle fields; or say between the extreme limits of about a fifth and a half of the steeper northerly dips. It is evident that the prevalence of unsymmetrical folds, the so-called "normal flexure," and of steep northerly dips is notably not so strong as was formerly supposed; and that it is very strikingly not so in the smaller folds of the Western Middle field and the larger ones of the Northern field.

Mr. Bailey Willis, in his very ingenious, richly illustrated and valuable memoir on "The Mechanics of Appalachian Structure," appended to the Thirteenth Annual Report of the United States Geological Survey, concludes, with apparent reason, that the Appalachian folds were caused by the contraction through cooling of the interior of the earth, a contraction, in itself very widespread, but in its visible effects principally concentrated within comparatively narrow limits; that the folding took place mainly in the space where the sediments of the Paleozoic sea had accumulated in great thickness and had de-

pressed the underlying support, undoubtedly plastic and yielding under so great a load, thereby forming a more or less decided trough filled with the Paleozoic rocks; and that the contraction pressed against the sides of the trough with force so irresistible as to crumple the rock-beds into folds. Of course, the contraction beneath the Appalachian region itself would be added to the much greater contraction of the wider spaces outside. He argues that the place of the fold is partly determined by the original inclination, or, as he calls it, the "initial dip" of the beds, as indicated by their varying thickness; and perhaps even seemingly slight irregularities of surface had still more influence on the place of the origin of folds than he claims. Besides, the Paleozoic rocks, by their very sinking, would in some small degree tend to crumple, and would probably originate folds, according, not merely to the composition, firmness and character of the beds and groups of beds, but to any unevenness there might be from previous erosion in the ancient floor, and to any lightening of the load from erosion of the more recent surface during the folding. He points out that as the pressure on the beds is mainly horizontal from the sides, folds depend on the existence of beds or groups of beds of firm or structural character, that is, coherent, stiff and strong enough ("competent," as he calls them) to transmit the pressure and to lift up the load of rock-beds above them; and he shows that the sidewise pressure would be resolved by any irregularity in the direction of the firm beds into forces partly parallel with them and partly radial to their curves; that the radial forces would always push from the concave side of the curve towards the convex side; and, moreover, that the increased downward-pressing weight of the further side of any fold raised would occasion the transmisssion of the force onwards, so as to raise other "consequent folds," as he calls them, beyond the original main fold. He shows also that the surface-length, or profile-length, "dip-length" as he calls it, of a fold, depends, in cases where the conditions are otherwise uniform, either on the stiffness of the structural group of beds or on the weight of superincumbent beds. In such cases, namely, the arch will, for a given load, necessarily be as long as the stiffness allows (that is, the curve of least resistance), and obviously cannot be longer. But the surface or profile-length of the curve

of a bed of given stiffness will be inversely proportioned to the load (that is, sharper and shorter, the greater the load). He argues, too, perhaps not quite clearly, that his "consequent" folds would be shorter than the original ones, but the single experimental result he cites seems rather to prove the contrary. The two shorter folds therein indicated, though subsequent, are not "consequent," in the sense of being transmitted through the original fold; for they are between that fold and the applied force, and may be shorter for other reasons, such as some slight unevenness or lack of homogeneity in the layers, or the increased nearness of the point of resistance by reason of the downward pressure of the original fold.

The shorter measure, then, of decidedly subordinate folds, within any space so limited that the conditions otherwise (including the superincumbent load) must be practically uniform, would seem clearly to be due to the fact that only a subordinate group of beds of less total stiffness is concerned. Also, at certain points in folds where special pressure exists, equivalent in its effect to increased load, narrower, subordinate folds may arise in subordinate groups of weaker beds confined between stiffer beds above and below. As the subordinate folds arise from the transmission of the pressure through the adjoining main folds, there would seem necessarily (under otherwise uniform conditions) to be throughout their length equal profile (or dip) length between the main and subordinate ones. But that, in the case or a rising or sinking ("pitching") anticlinal, would cause the subordinate folds riding upon it to have their axes not horizontally parallel with the main axis; so that a subordinate fold would descend the flank of a sinking anticlinal. It has long been known that the subordinate folds riding upon a main one are not parallel to it. Lesley, for example, alluded to the fact in his *Manual of Coal and its Topography*, 1856, page 185, as follows: "The beds descend at all angles, even more than vertical, thrown over on their backs—snapped and the edges slipped past each other—crushed by small rolls running obliquely through the sides of the greater anticlinals, each one of which carries a dozen small ones on its back," etc. Materials are not at hand to prove conclusively that the obliqueness is in the direction and of the amount just now suggested, though in a certain case, partially worked out, it would seem to be so.

Every fault shown in the cross-sections (except N. A. F. IIA., B, the 4th on Plate VI.) is of the class called longitudinal faults, overlaps, or reversed faults, and appears to be the result of the extreme folding and overstraining of one of these subordinate folds, riding upon a northerly or a southerly dip in about an equal number of cases (17 northerly to 16 southerly). In 5 cases, to be sure, the faults occur very close to the bottom of main basins; but even there, apparently, the form is taken of subordinate basins and saddles, and the main ones are not inverted, but are virtually symmetrical or nearly so. In all the 31 cases where the dip of the faults is clear, it is overturned with one or two possible exceptions. The most decided exception seems to be that of Plate VI., N. A. F. IIA., B. In 24 cases out of the 31 the resulting overturned dip is southerly. The upthrow in every case is on the side towards which the fault dips, except in 1 case; and except possibly in 2 more, if the dip be not reckoned as reversed. The throw of the fault is up to the south in 24 cases out of 33. Of the 9 others the dip of the main fold is northerly in 5 cases, about an equal division. Of the 28 faults where the "stratigraphic throw" (that is, the shortest distance from a layer on one side of the fault to the same layer on the opposite side) can be somewhat closely measured, it varies from 10 feet to about 160 feet, and averages about 62 feet, and the displacement (of the two ends of a layer along the plane of the fault) in the same way varies, in 26 sections where it is pretty clear, from 20 feet up to perhaps 240 feet and averages about 72 feet.

As it is evident, then, that the faults arise from the folding of subordinate groups of beds between stiffer ones, the extent of the faults depends like that of the subordinate folds on the firmness of those subordinate groups, and cannot in the same region much exceed the limits observed in these numerous cases. That may serve as a practical guide of some value in regard to the possible extent of faults in the region. For it would plainly be absurd to imagine that faults exist of a size wholly disproportionate to these, say with a stratigraphic throw very much greater than any that has been observed in the whole region, or a displacement very far more extensive than has been observed anywhere there. For example, it would be wholly absurd to imagine a fault with a stratigraphic throw of

250 or 300 feet, such as would be necessary for identifying a coarse conglomerate with the Pottsville conglomerate (No. XII) instead of with the conglomerate that frequently occurs above the Mammoth coal-bed, after the manner pointed out in these *Transactions*, vol. xxi., p. 713, as occurring at a number of places, and particularly at one, half a dozen miles west of Tamaqua; where, however, the topography as well as ample natural and artificial geological exposures have independently fully disproved the old erroneous conjecture of a great fault. Such an isolated fault of gigantic size would in any case be an impossibility in the midst of a region where numerous observations show that the size of faults is limited to much smaller dimensions; for the extent of the faults depends on the stiffness and strength of the beds and the original weight of the overlying beds, and in these respects there is no essential difference within the small space of one limited region.

We may conclude, then, that steep northerly dips in the Pennsylvania anthracite-region are much less prevalent than was formerly supposed; that nearly half the basins and saddles are about symmetrical; and that nearly three-fourths of the subordinate ones are so in the Western Middle field; but that less than a quarter of the main ones are so in the Southern field. Again, that the subordinate folds throughout the region are confined to subordinate groups of beds of inferior firmness, and are not parallel to the main folds, but probably at uniform profile-distances from the main axes, so as to descend the flank of a sinking anticlinal. Further, that the faults are almost invariably longitudinal or reversed faults, occasioned by the overstraining of subordinate folds, and corresponding in three-fourths of the cases to an overturned southerly dip, with the upthrow to the south; that such broken subordinate folds, whether dipping southerly or northerly, ride in equal number on the northerly-dipping and southerly-dipping sides of the main folds; that the stratigraphic throw averages only about 62 feet, and never exceeds 160 feet; and that the displacement averages 72 feet, and never exceeds 240 feet.

PLATE I.

CROSS-SECTIONS
OF
FOLDS AND FAULTS
OF ACTUAL WORKINGS IN
ANTHRACITE BEDS,
COMPILED BY
BENJAMIN SMITH LYMAN
FROM THE
CROSS-SECTION SHEETS
OF THE
PENNSYLVANIA GEOLOGICAL SURVEY.
FEB., 1895

THE HORIZONTAL AND VERTICAL SCALE OF THE SECTIONS IS 500 FEET TO AN INCH, UNLESS OTHERWISE STATED.

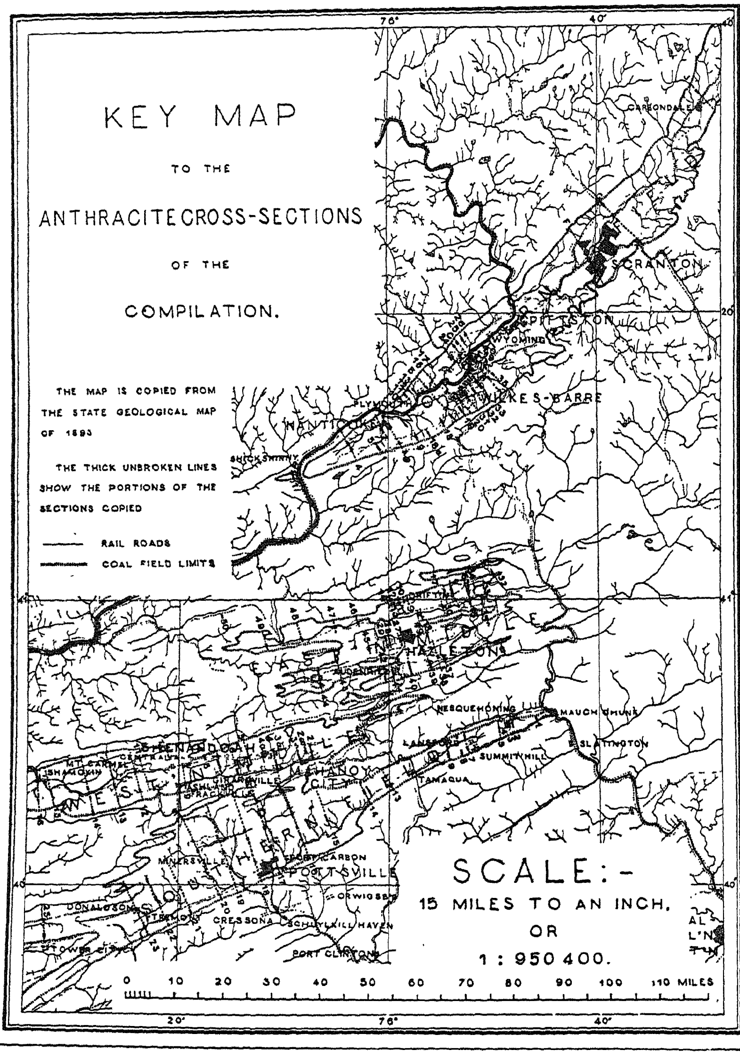
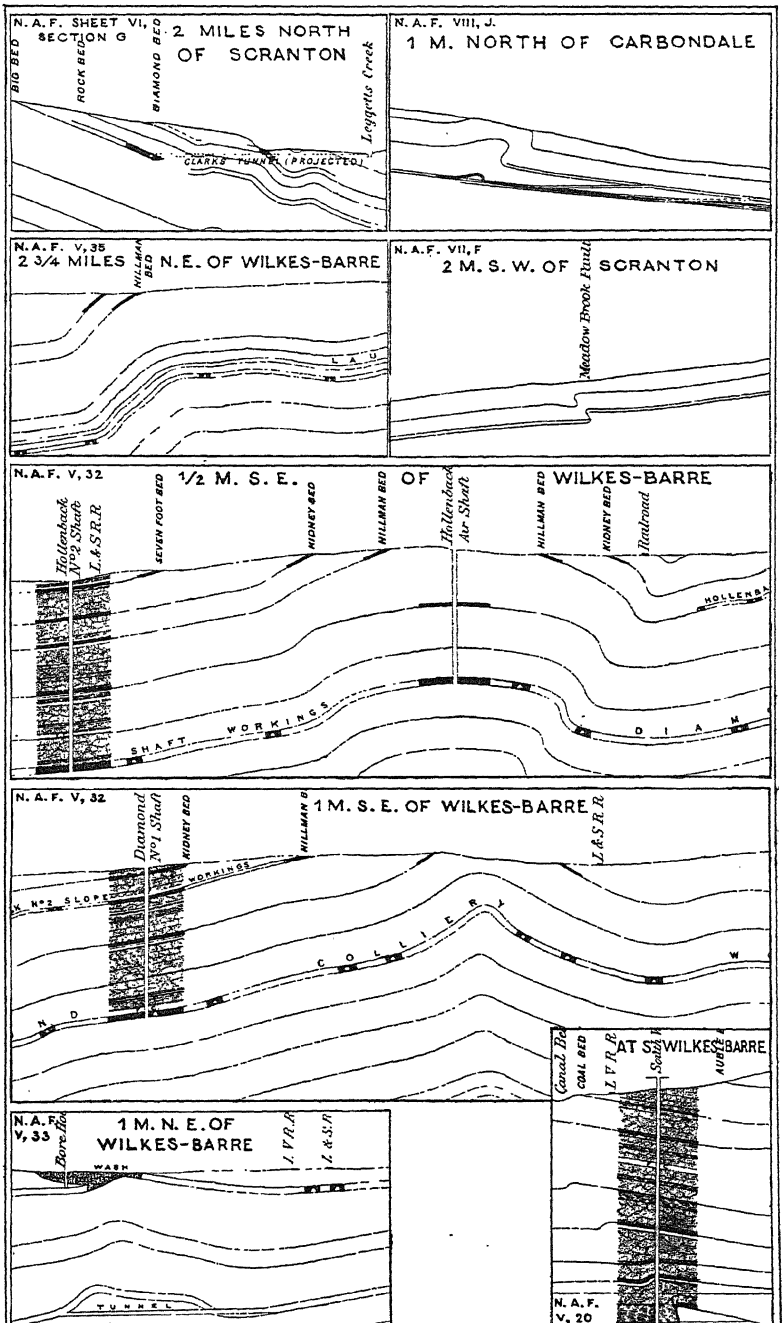
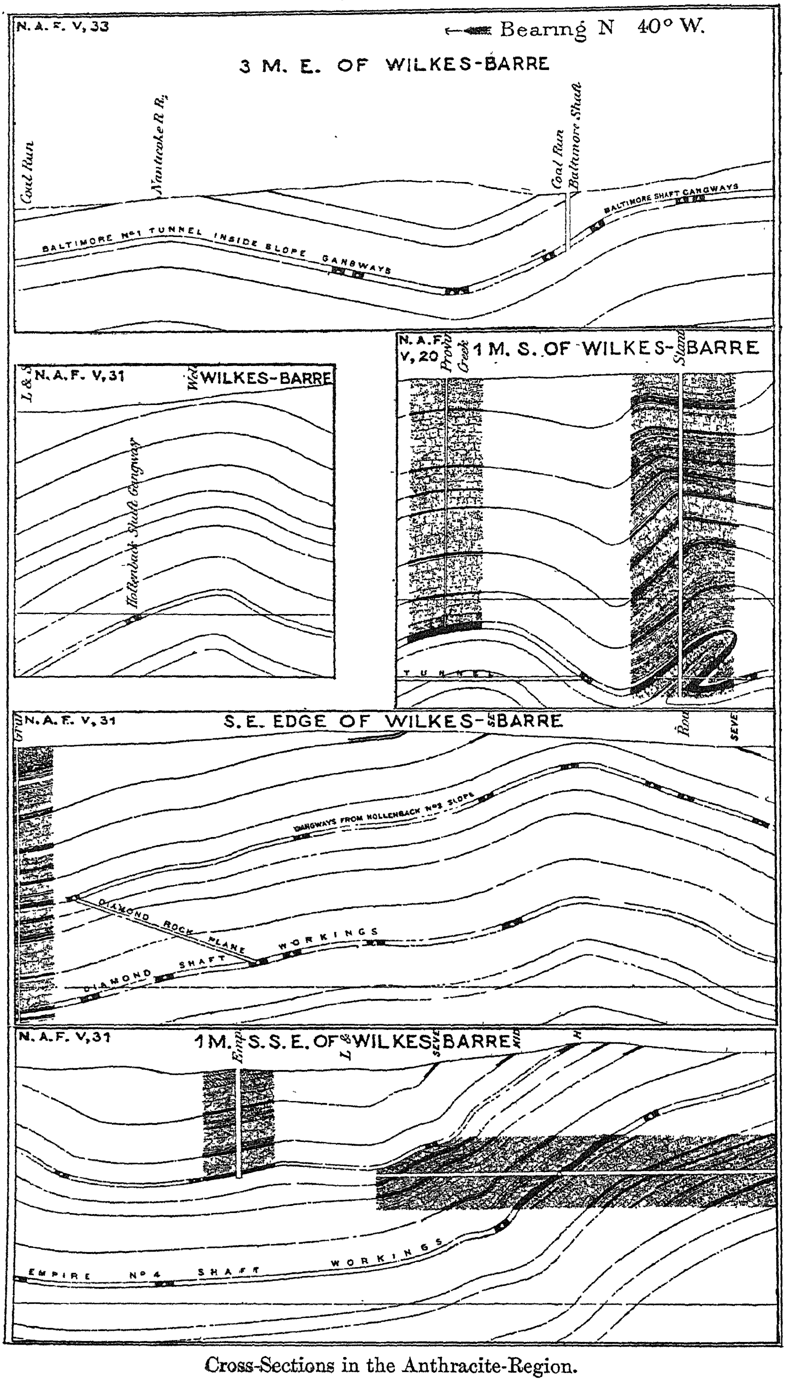


PLATE II.



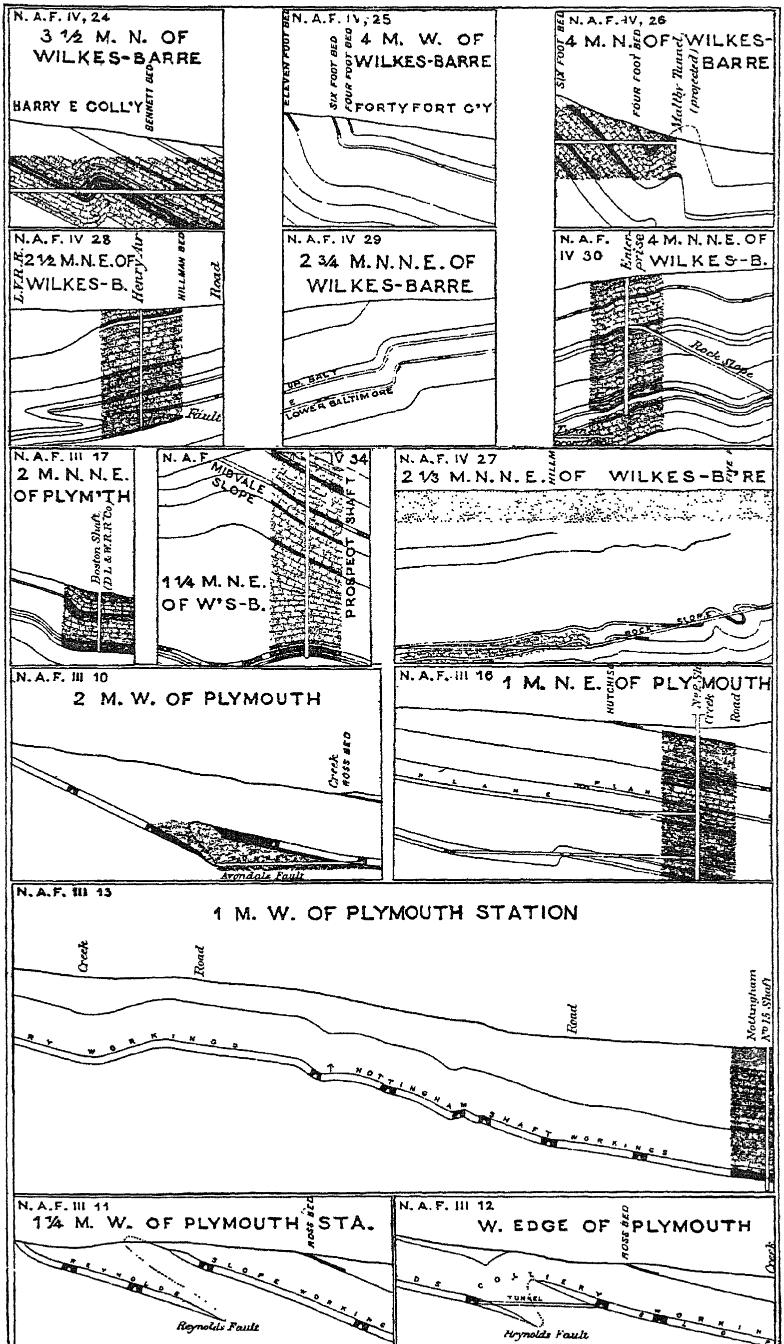
Cross-Sections in the Anthracite-Region.

PLATE III.



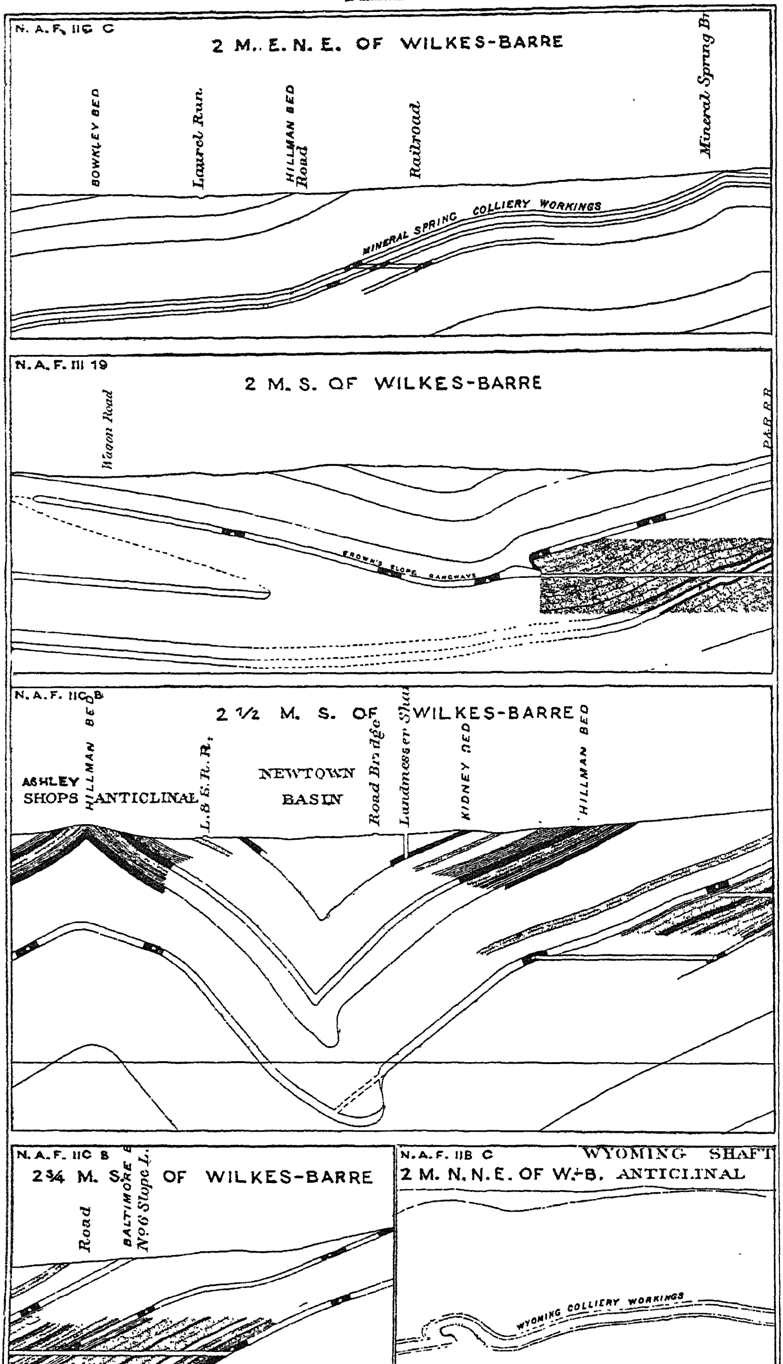
Cross-Sections in the Anthracite-Region.

PLATE IV.



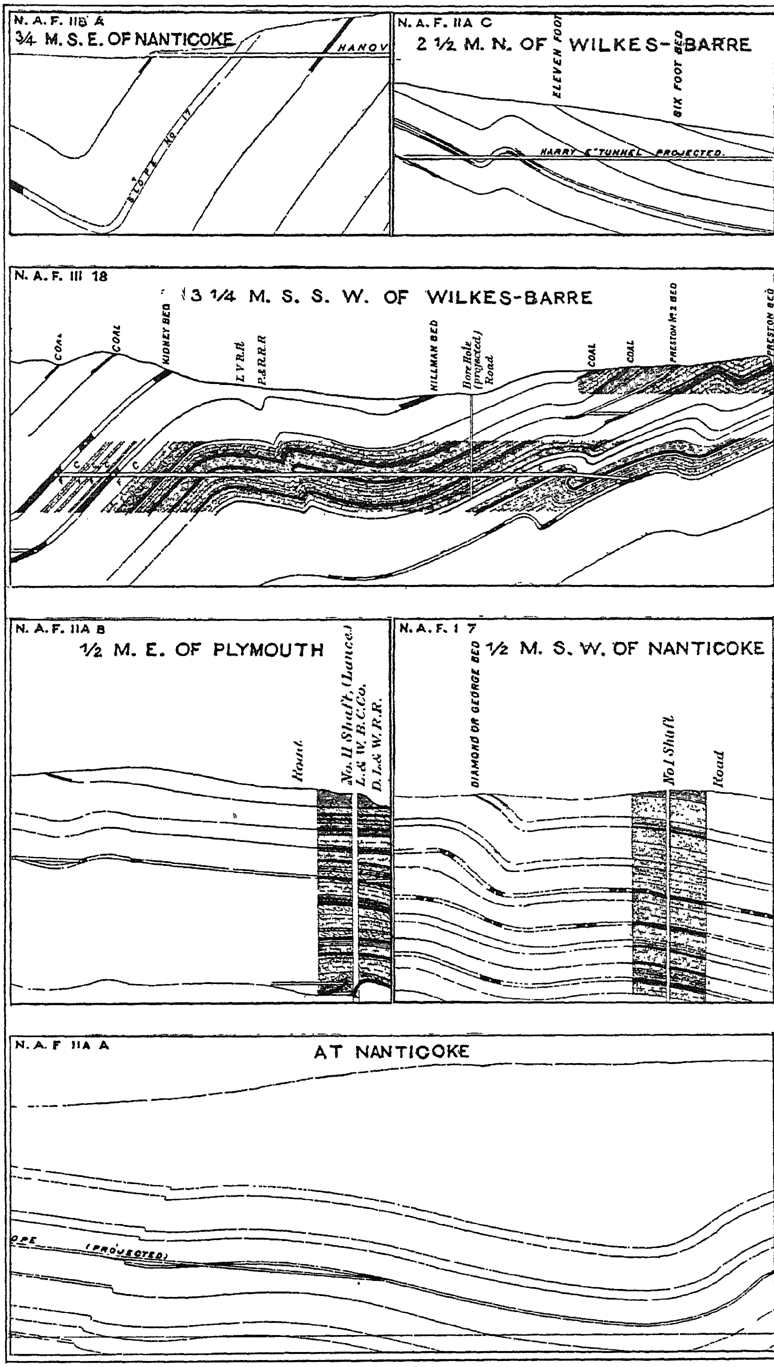
Cross-Sections in the Anthracite-Region.

PLATE V.



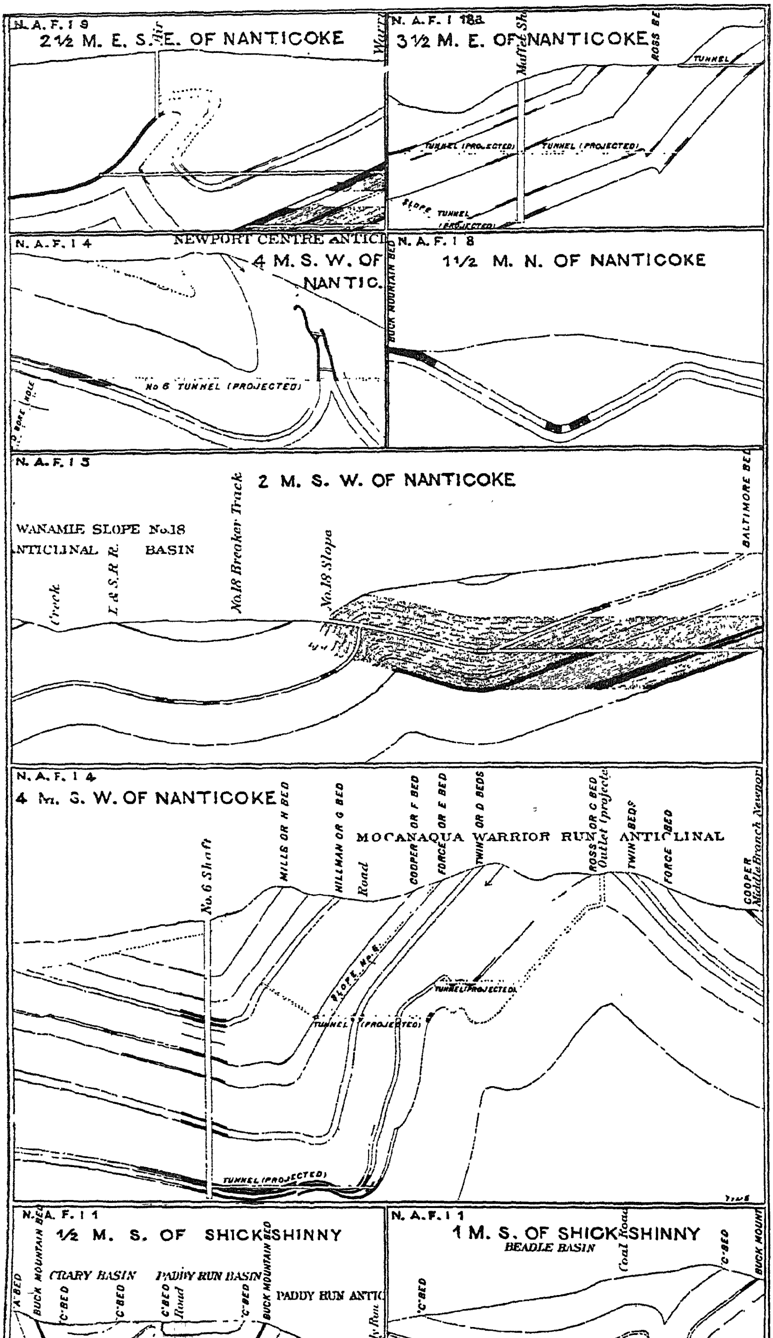
Cross-Sections in the Anthracite-Region.

PLATE VI.



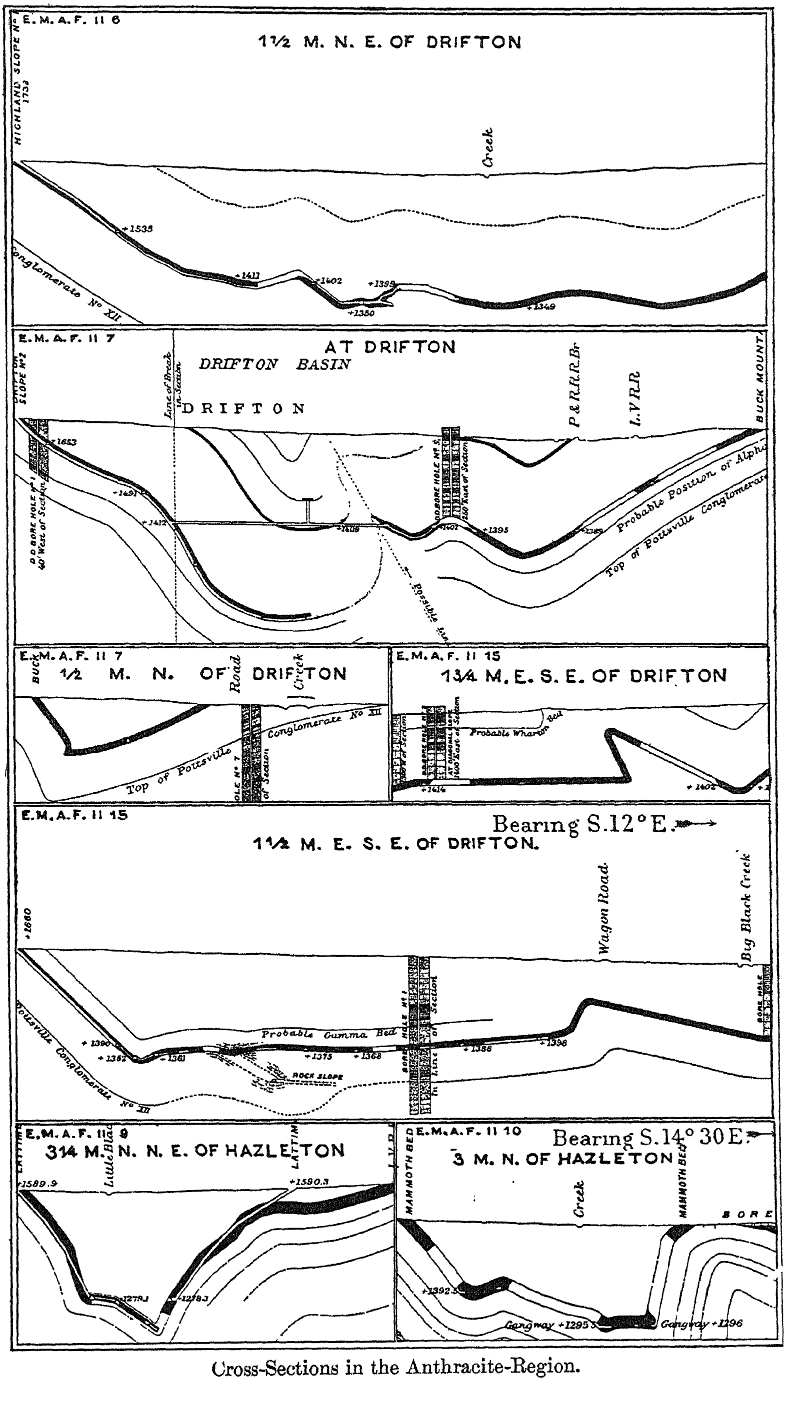
Cross-Sections in the Anthracite-Region.

PLATE VII.



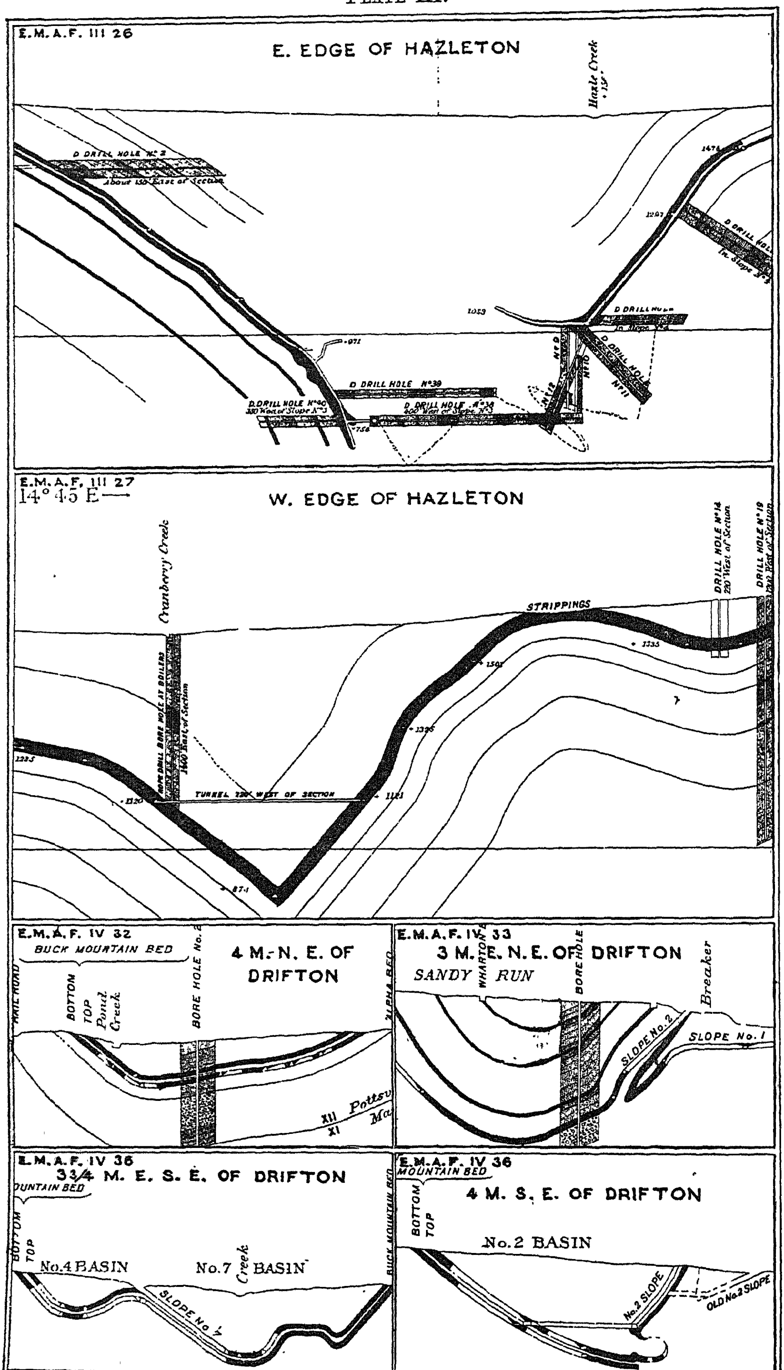
Cross-Sections in the Anthracite-Region.

PLATE VIII.



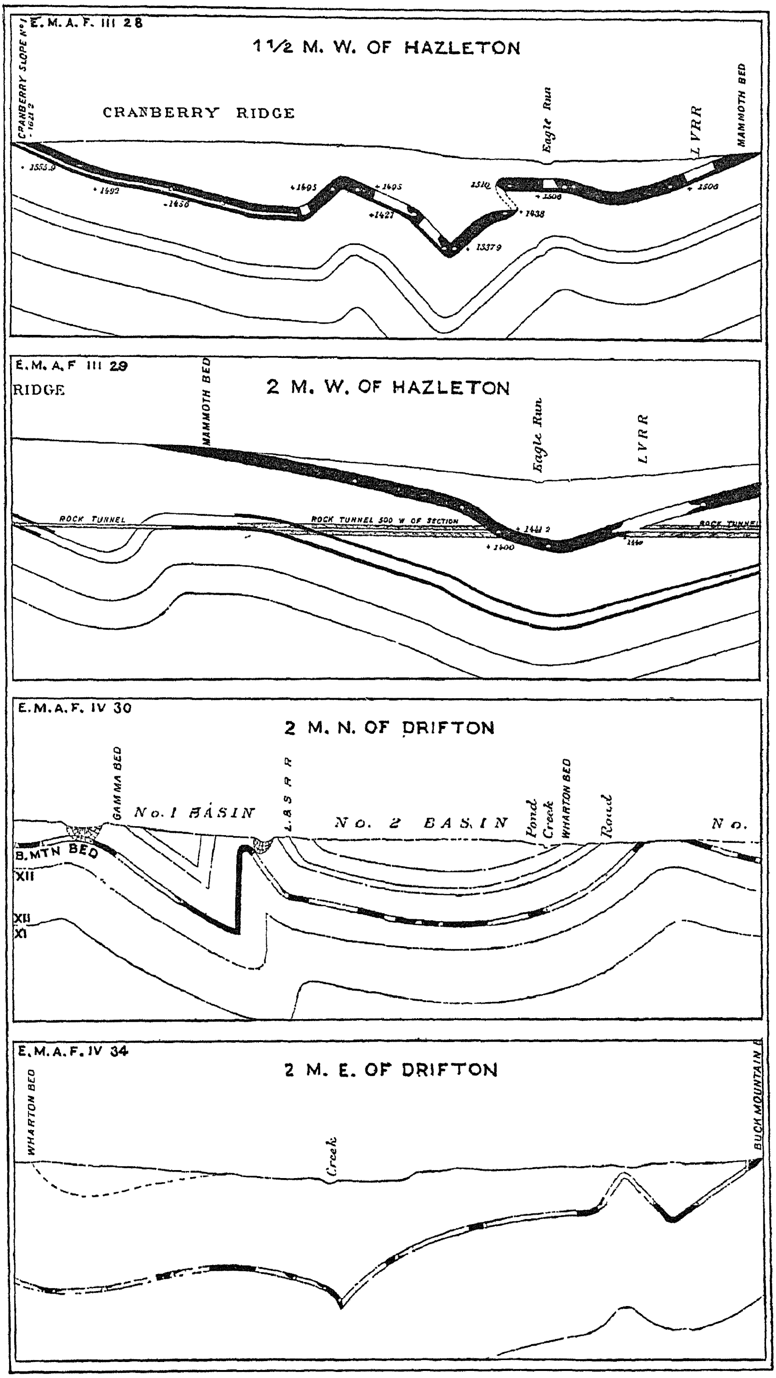
Cross-Sections in the Anthracite-Region.

PLATE IX.



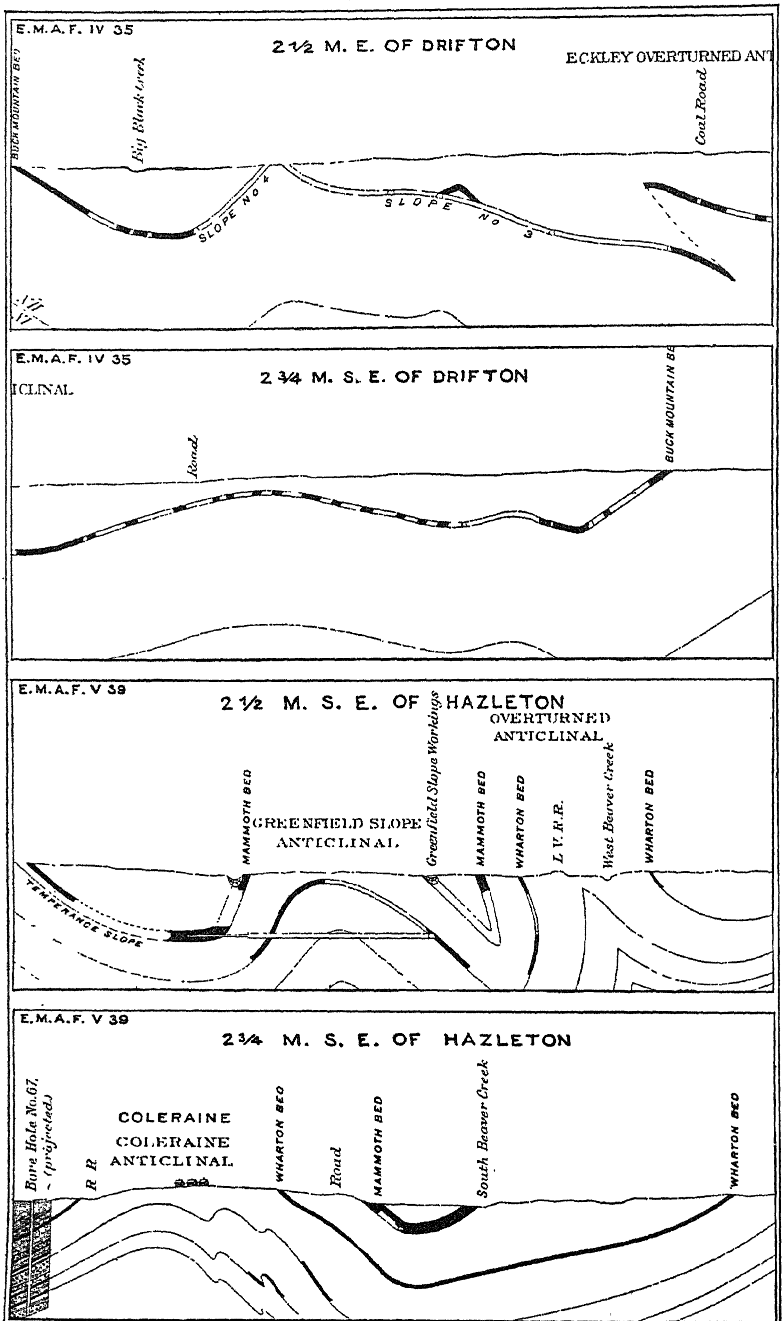
Cross-Sections in the Anthracite-Region.

PLATE X.



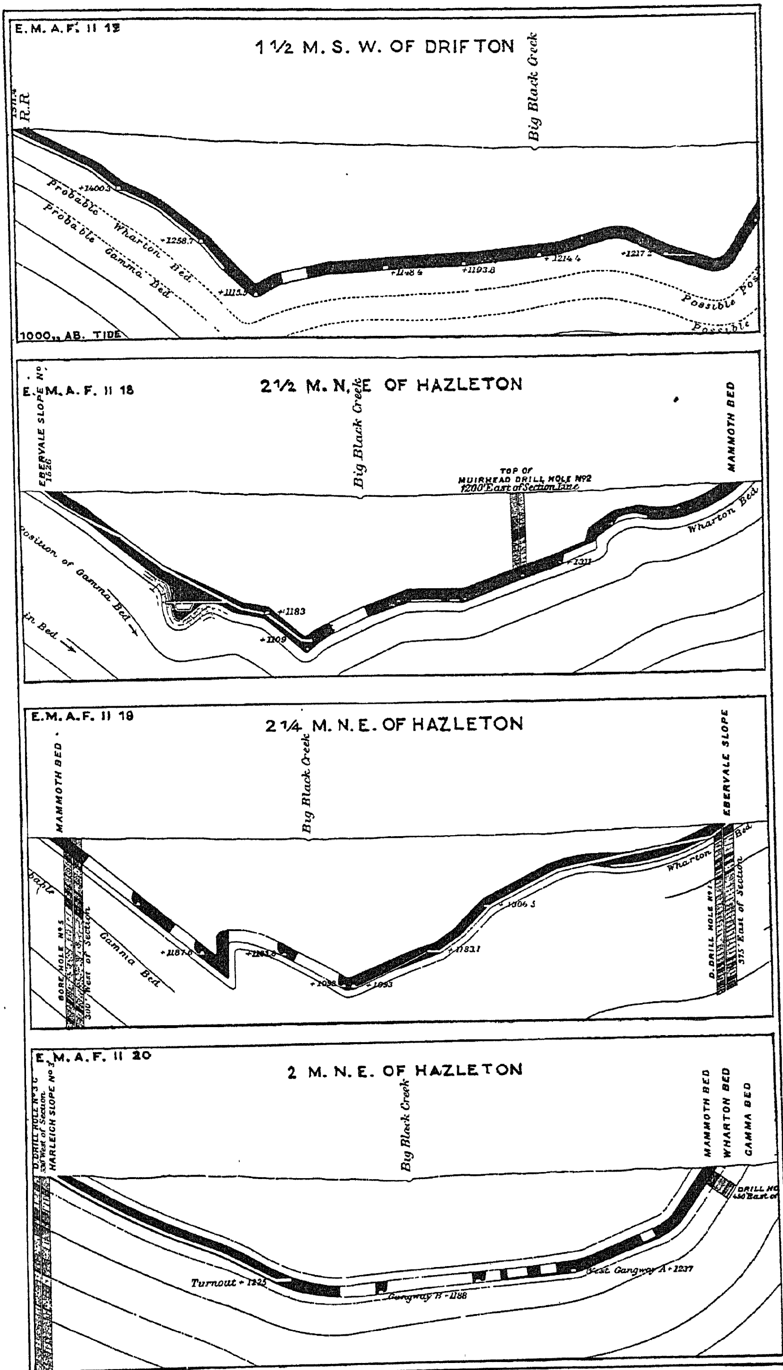
Cross-Sections in the Anthracite-Region.

PLATE XI.



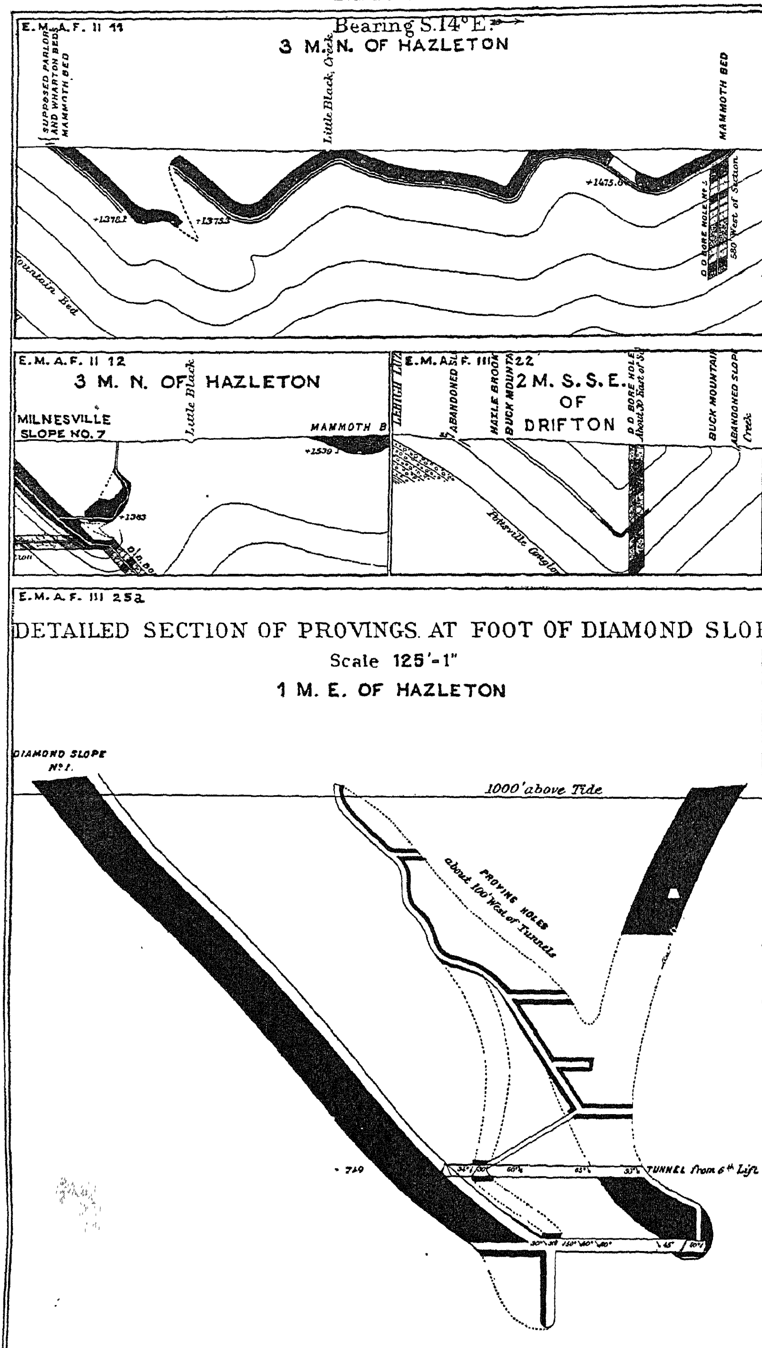
Cross-Sections in the Anthracite-Region.

PLATE XII.



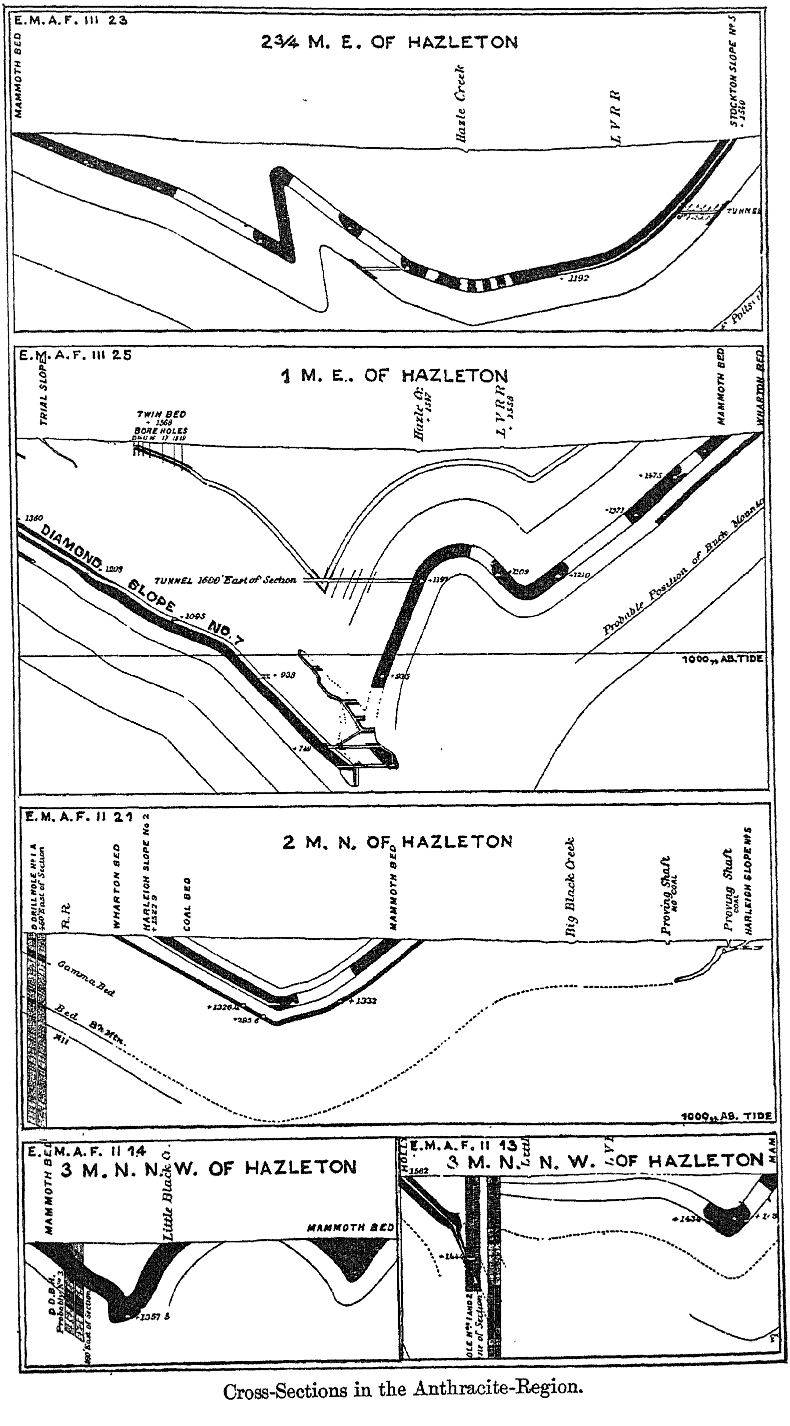
Cross-Sections in the Anthracite-Region.

PLATE XIII.



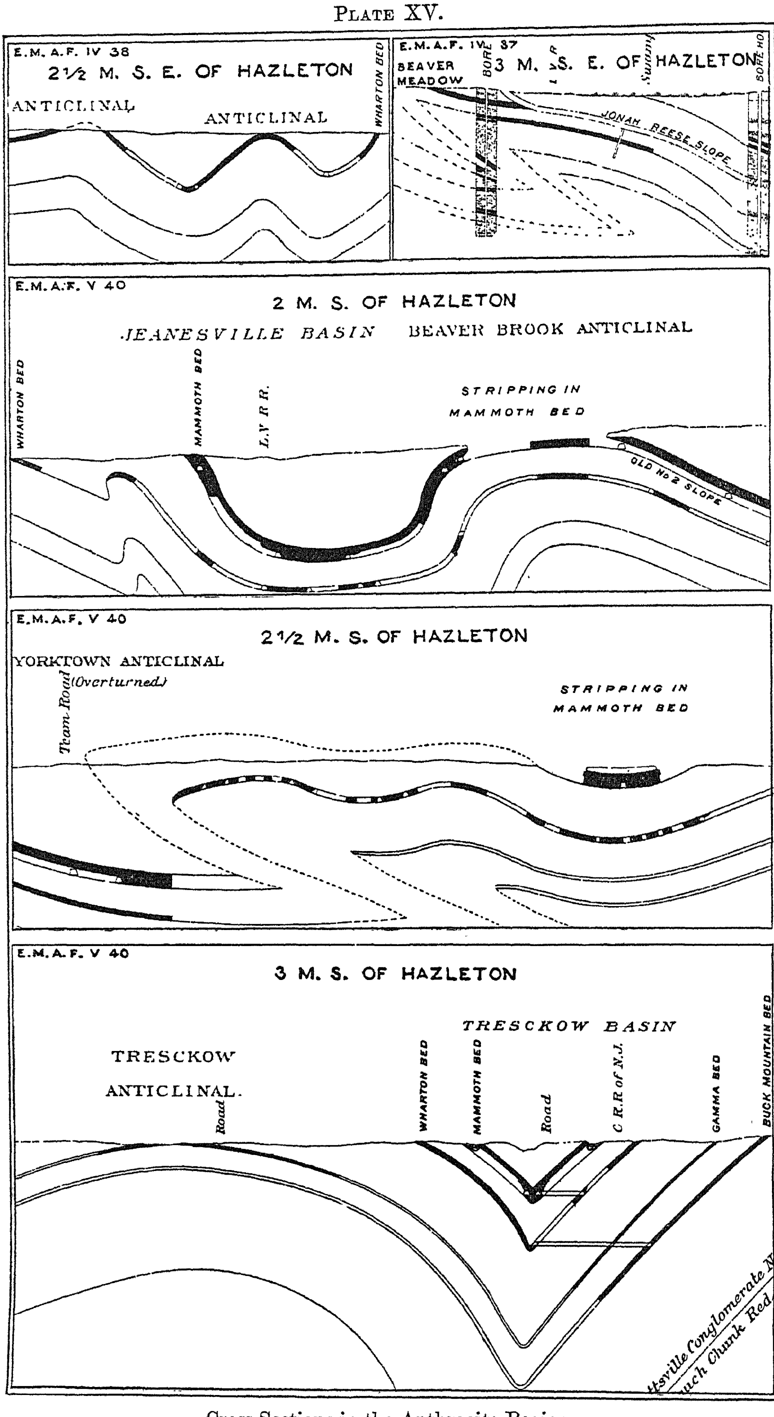
Cross-Sections in the Anthracite-Region.

PLATE XIV.



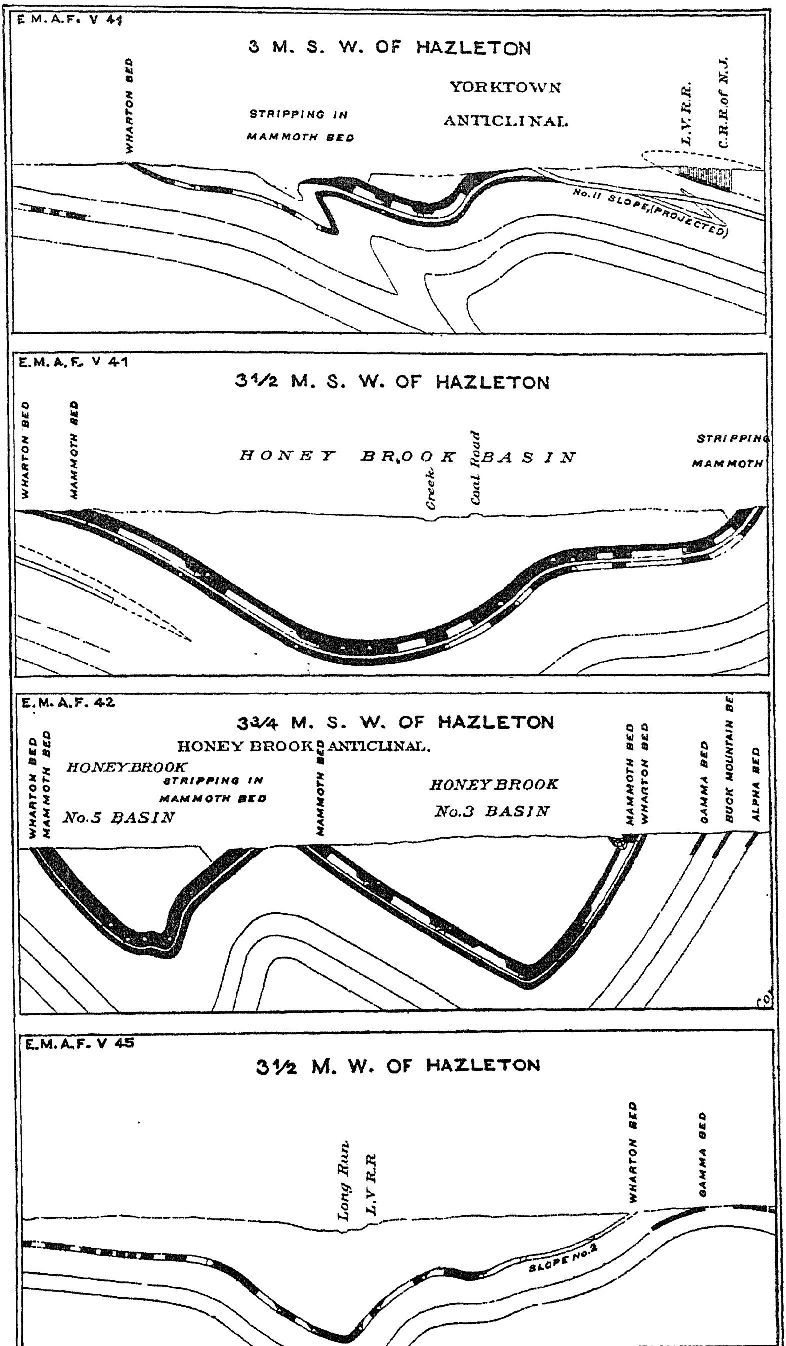
Cross-Sections in the Anthracite-Region.

PLATE XV.



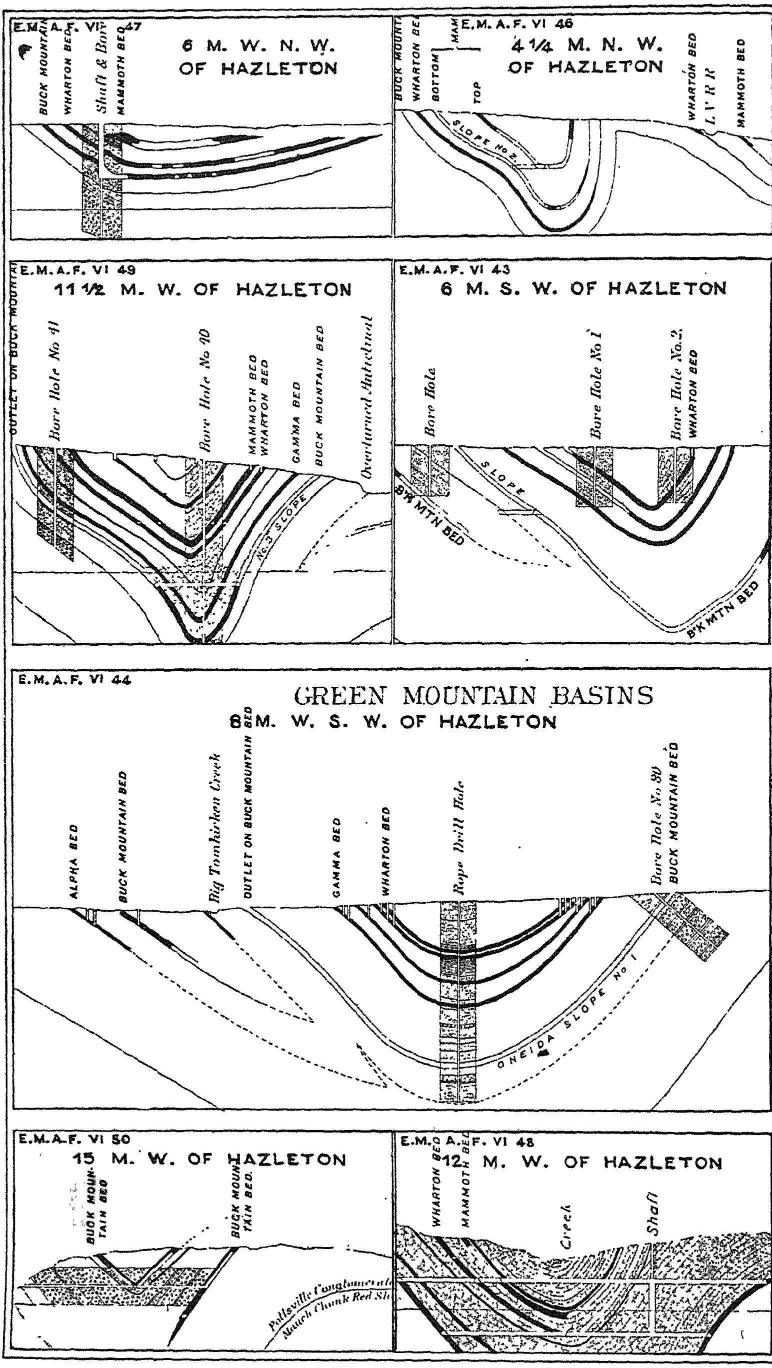
Cross-Sections in the Anthracite-Region.

PLATE XVI



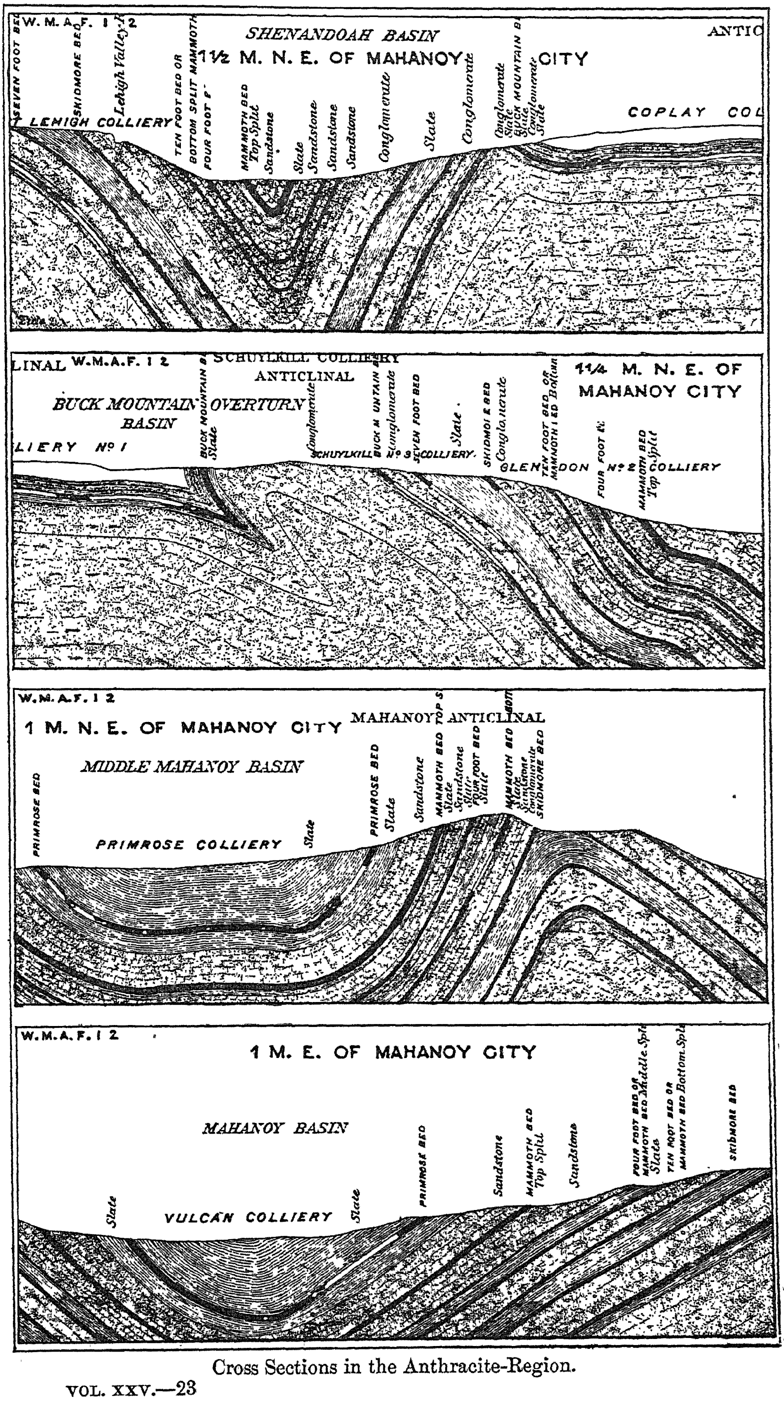
Cross-Sections in the Anthracite-Region.

PLATE XVII.



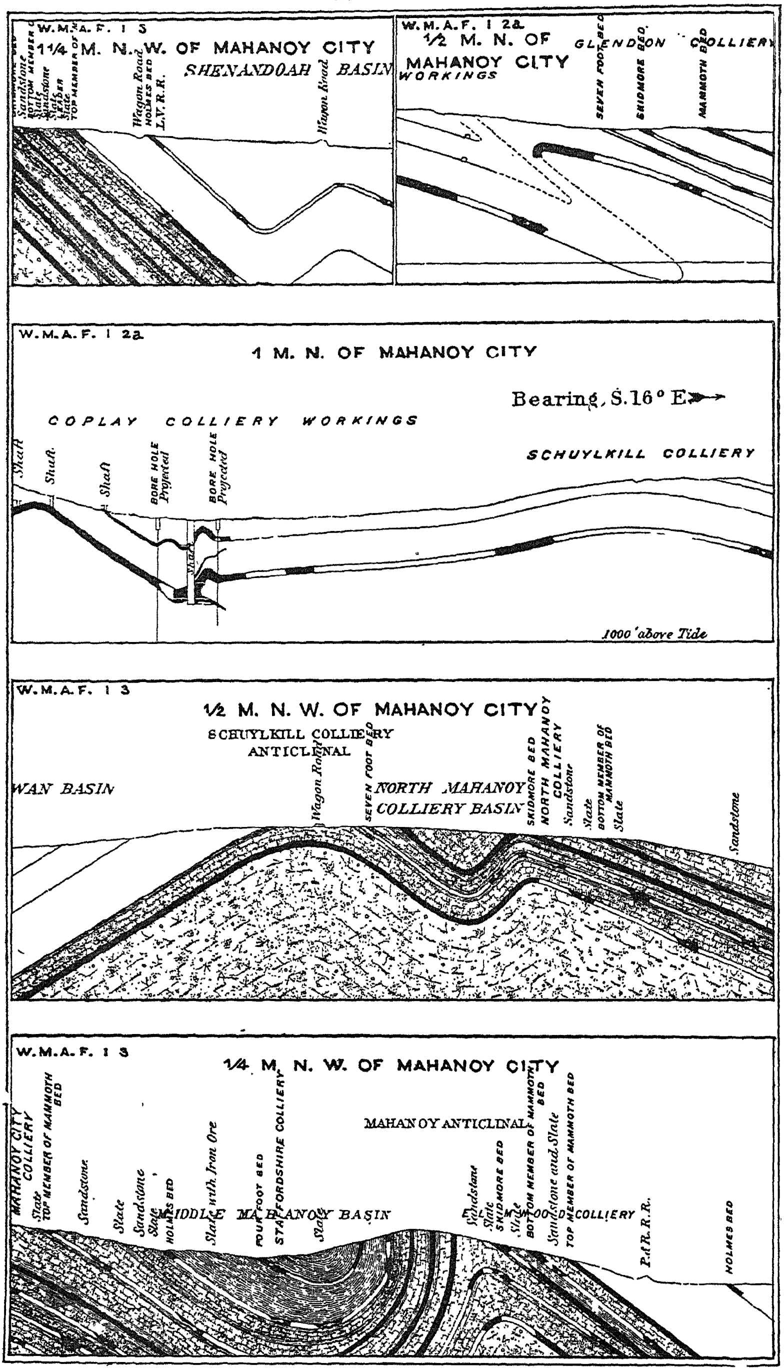
Cross-Sections in the Anthracite-Region.

PLATE XVIII.



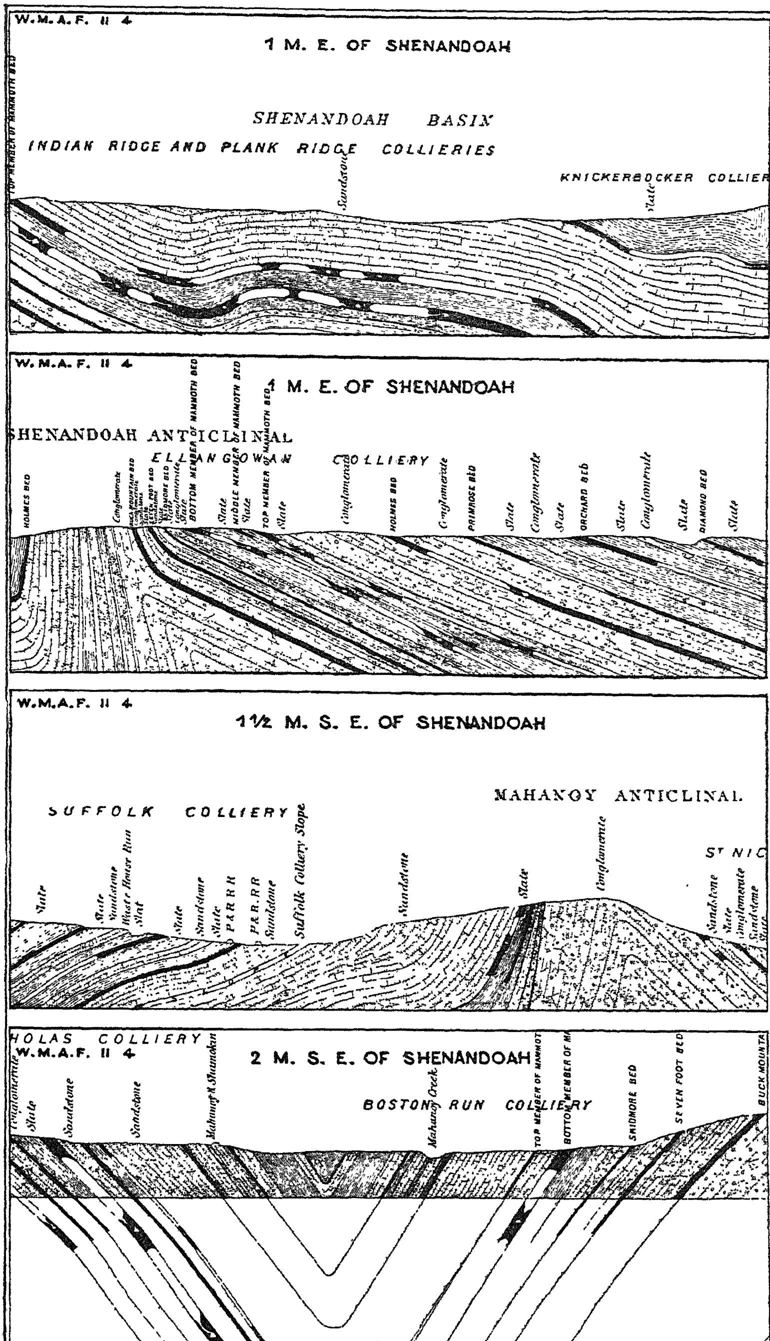
Cross Sections in the Anthracite-Region.

PLATE XIX.



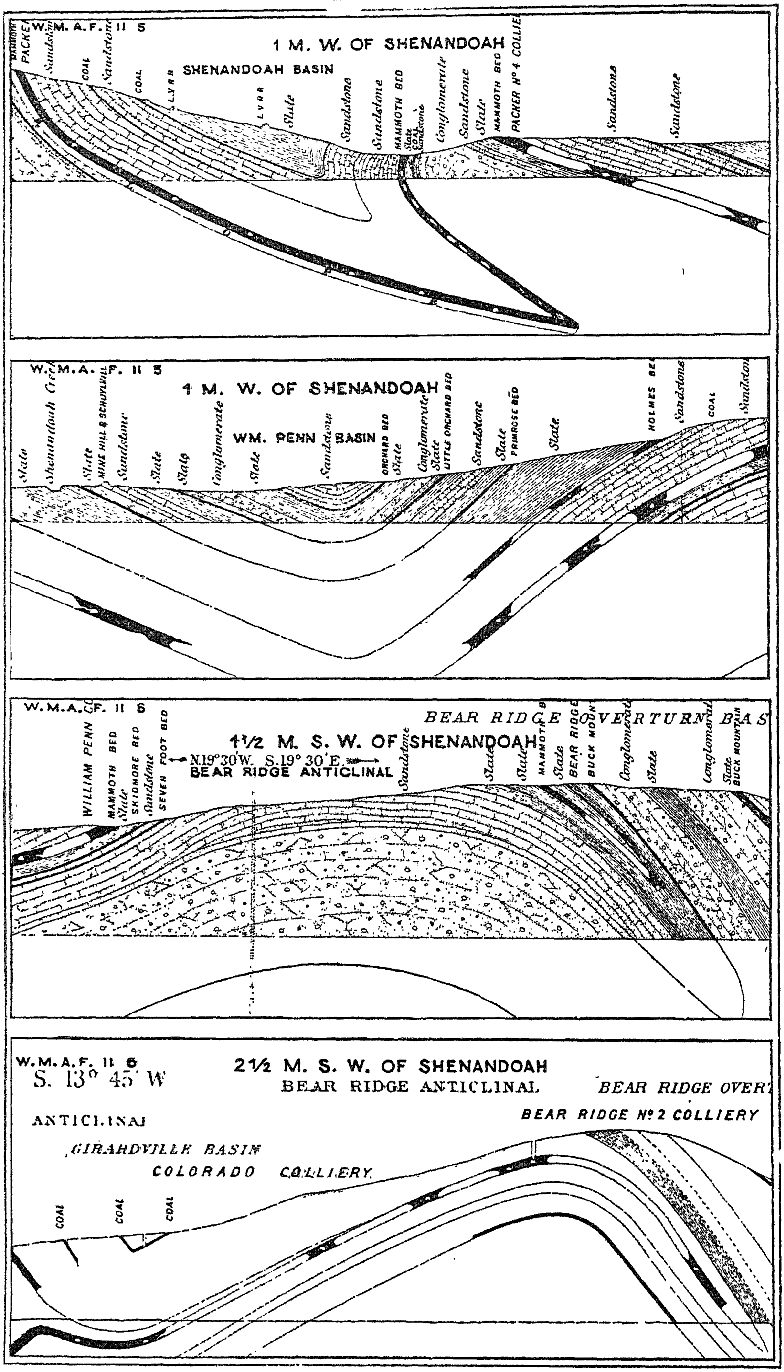
Cross-Sections in the Anthracite-Region.

PLATE XX.



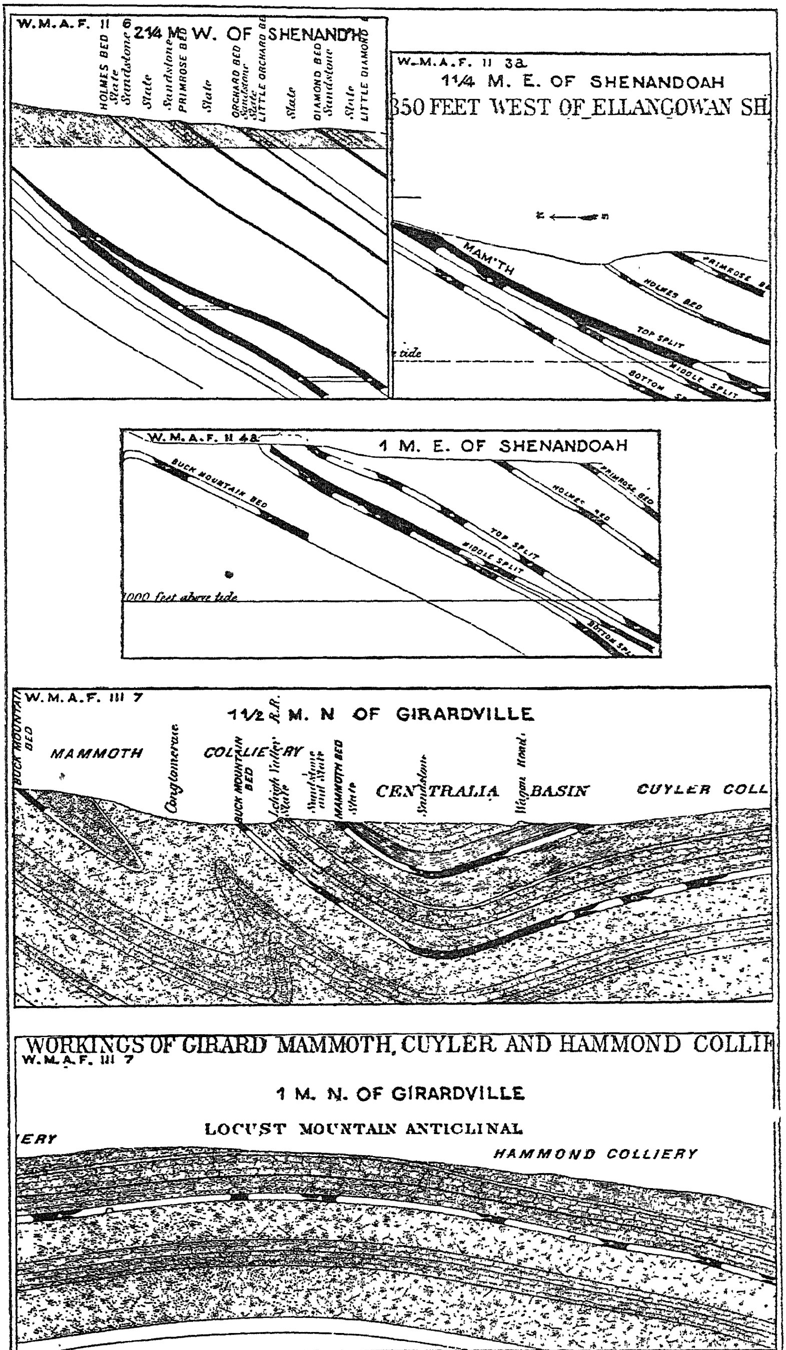
Cross-Sections in the Anthracite-Region.

PLATE XXI.



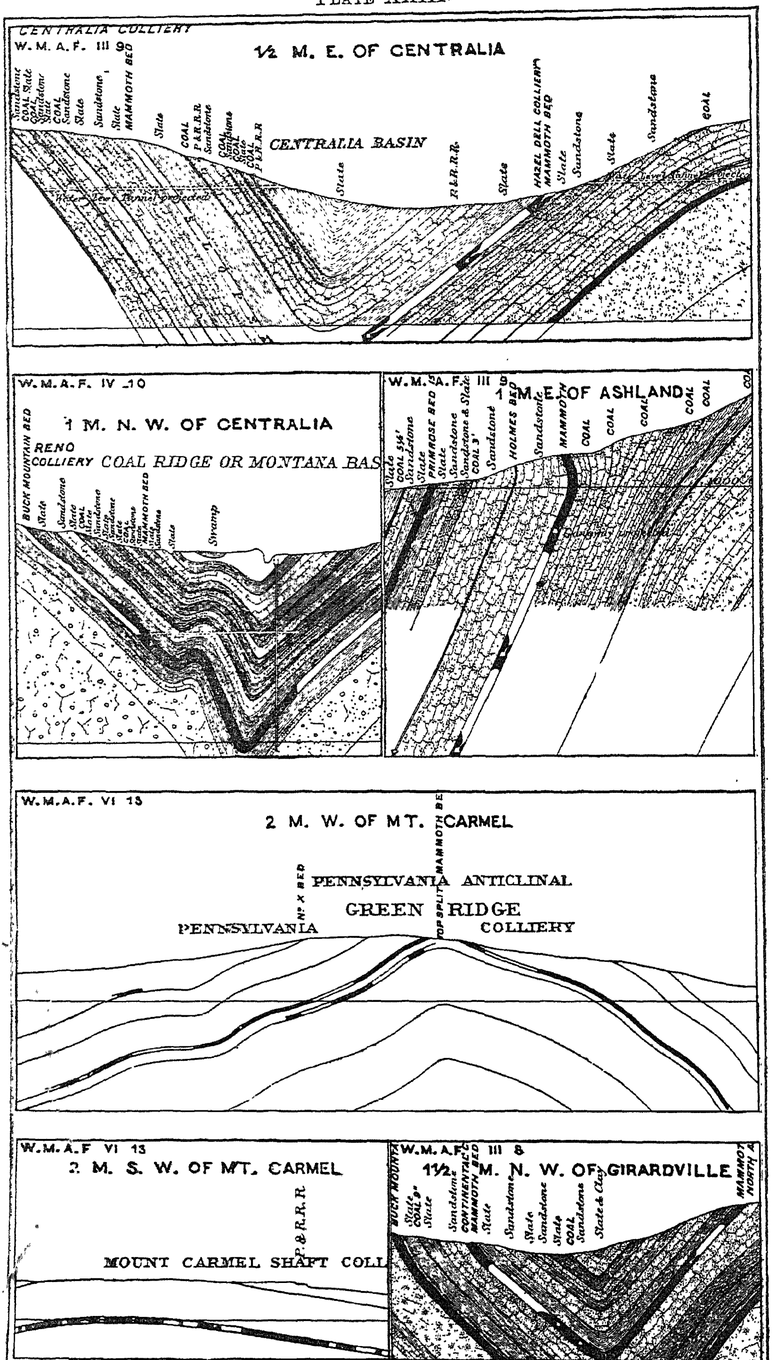
Cross-Sections in the Anthracite-Region.

PLATE XXII.



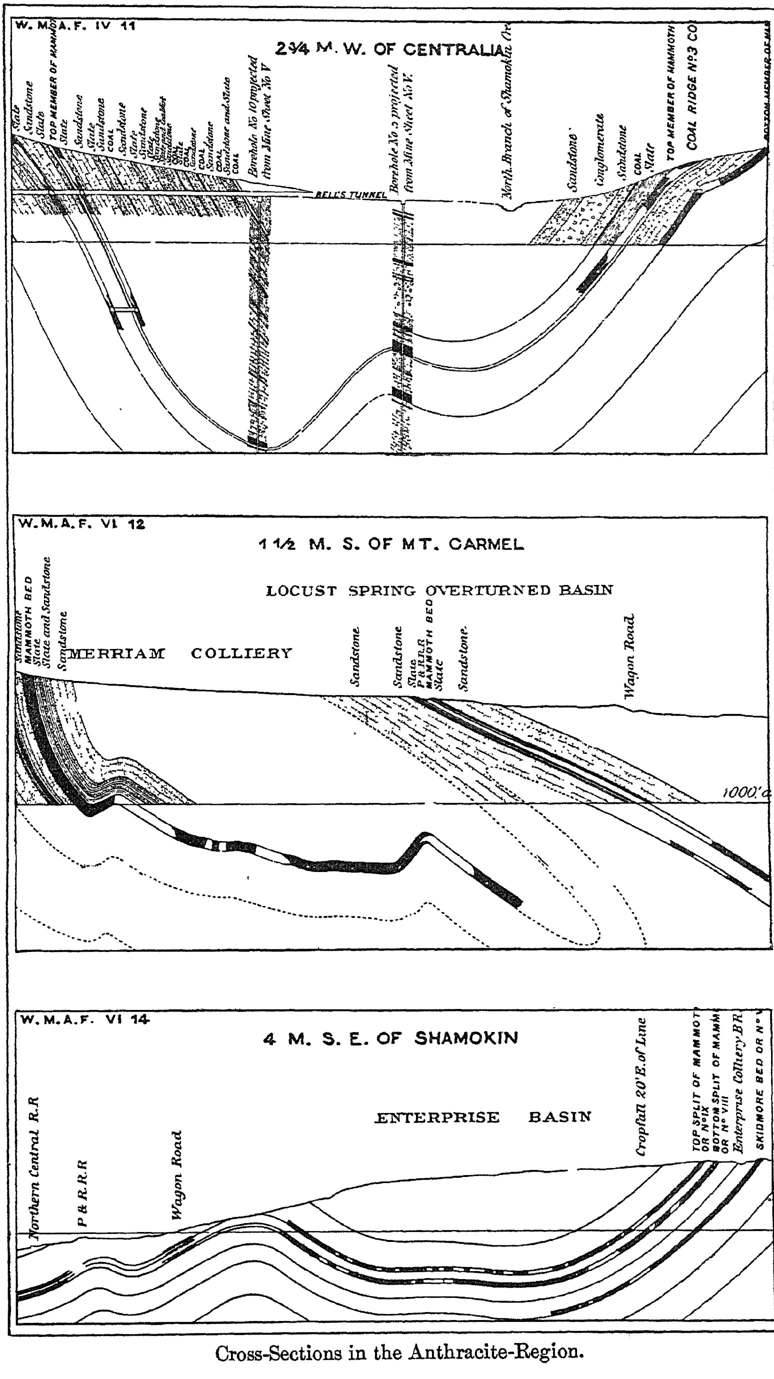
Cross-Sections in the Anthracite-Region.

PLATE XXIII.



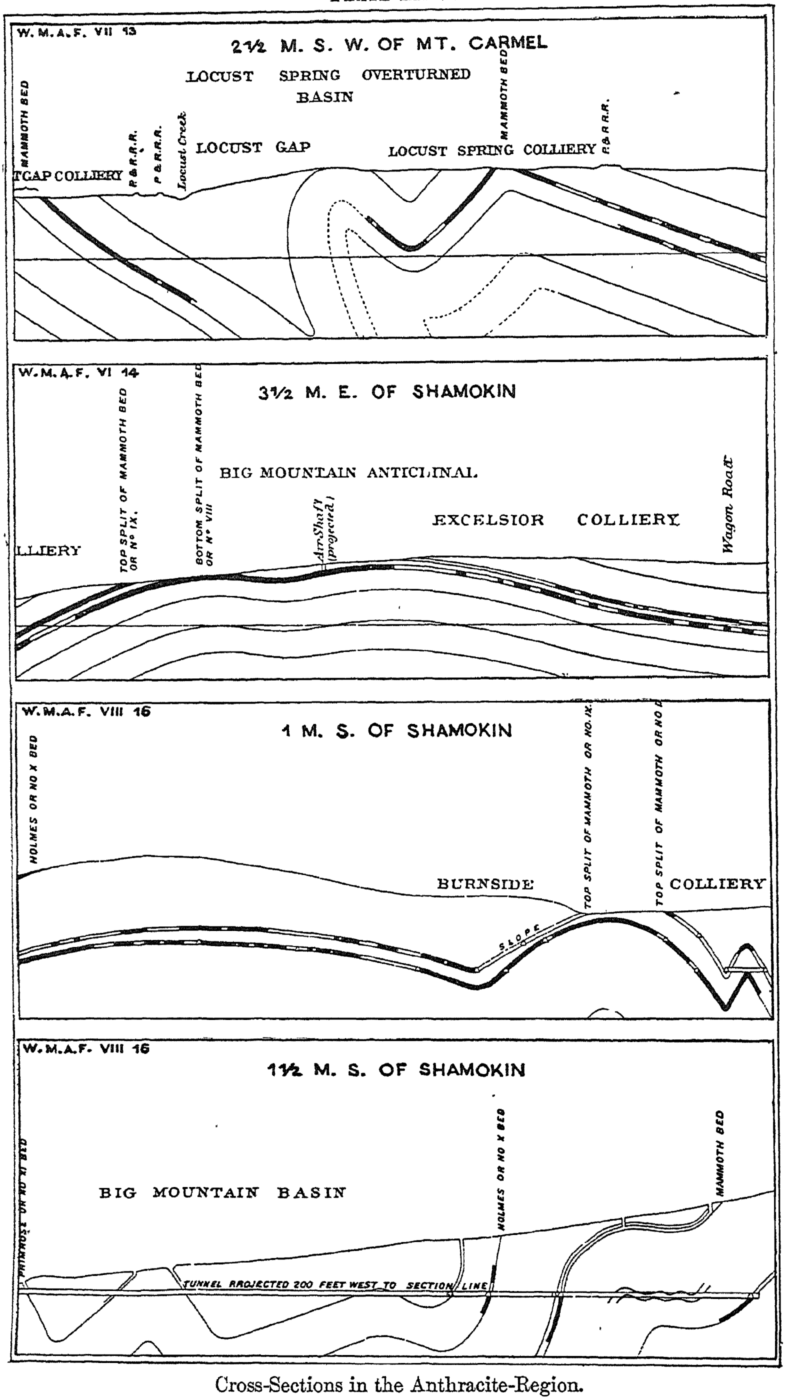
Cross-Sections in the Anthracite-Region.

PLATE XXIV.



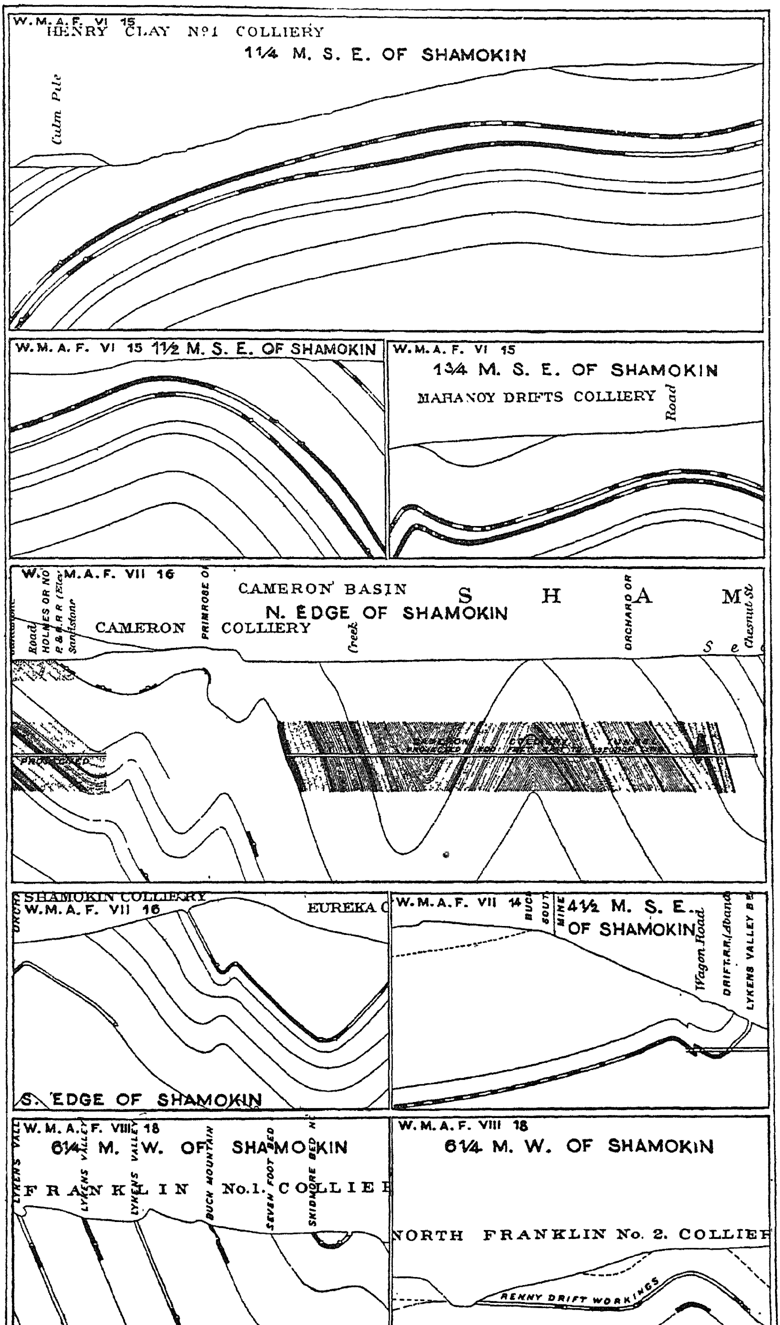
Cross-Sections in the Anthracite-Region.

PLATE XXV.



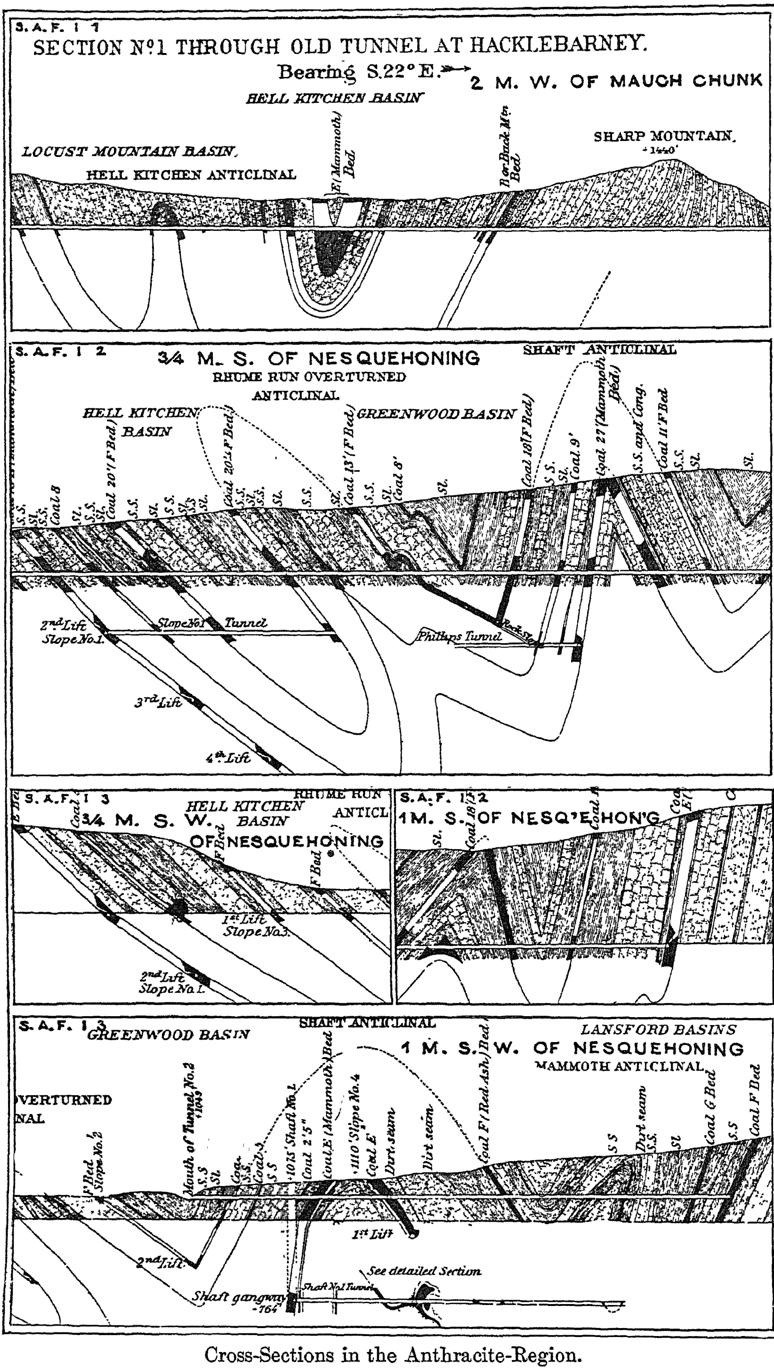
Cross-Sections in the Anthracite-Region.

PLATE XXVI.



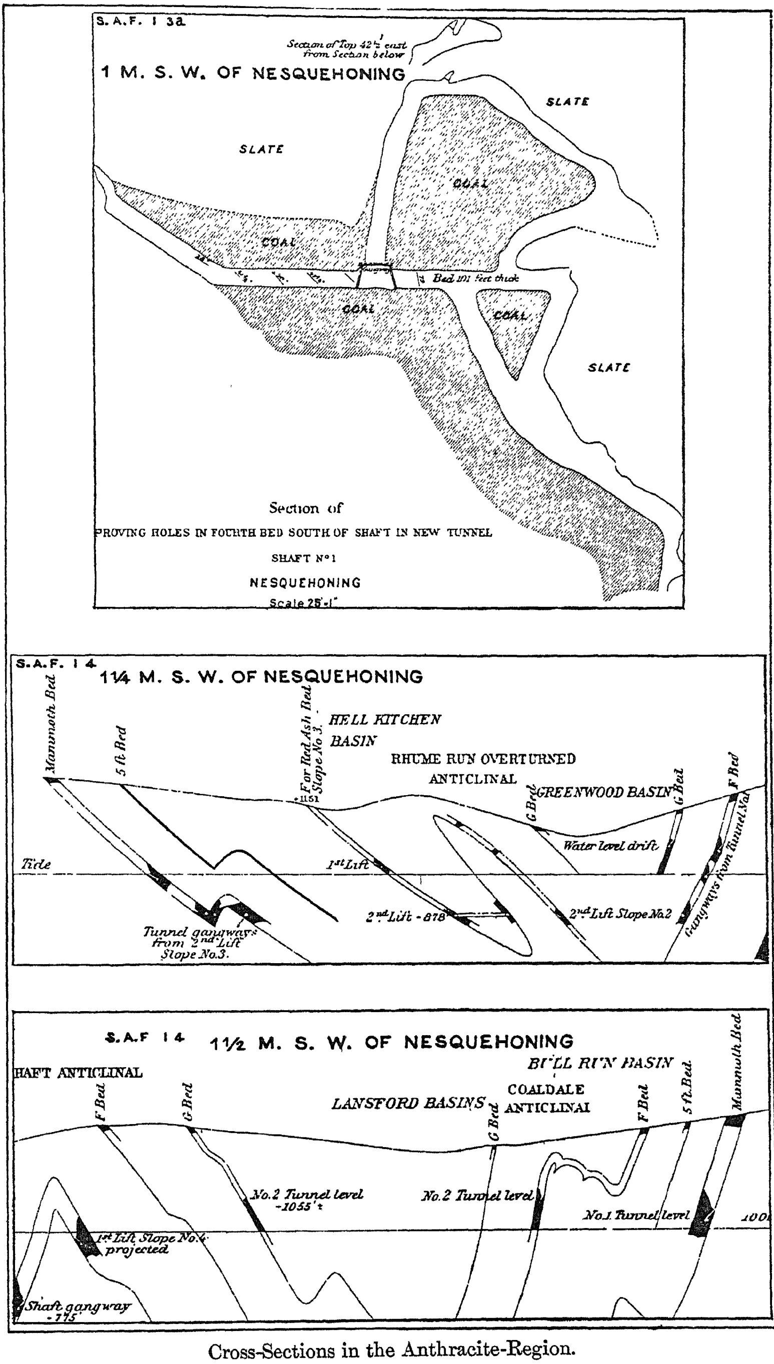
Cross-Sections in the Anthracite-Region.

PLATE XXVII.



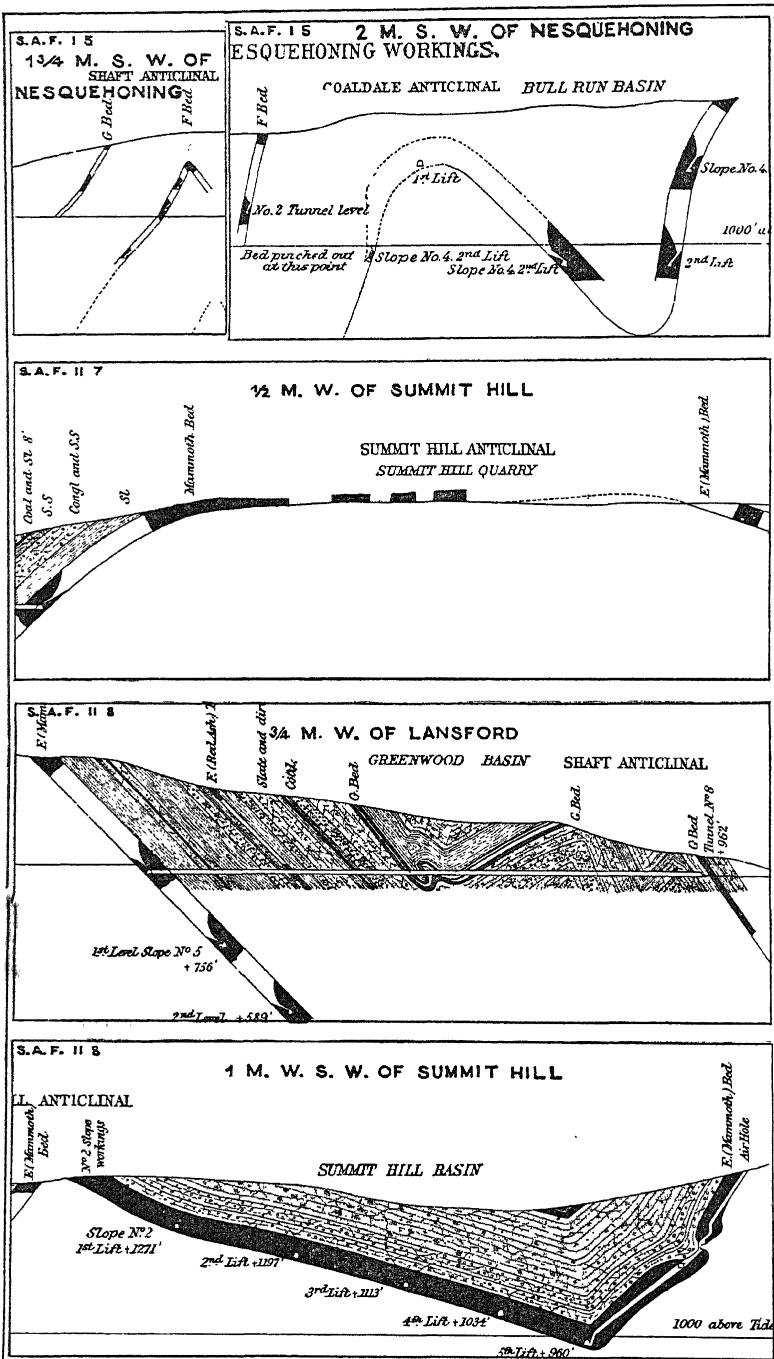
Cross-Sections in the Anthracite-Region.

PLATE XXVIII.



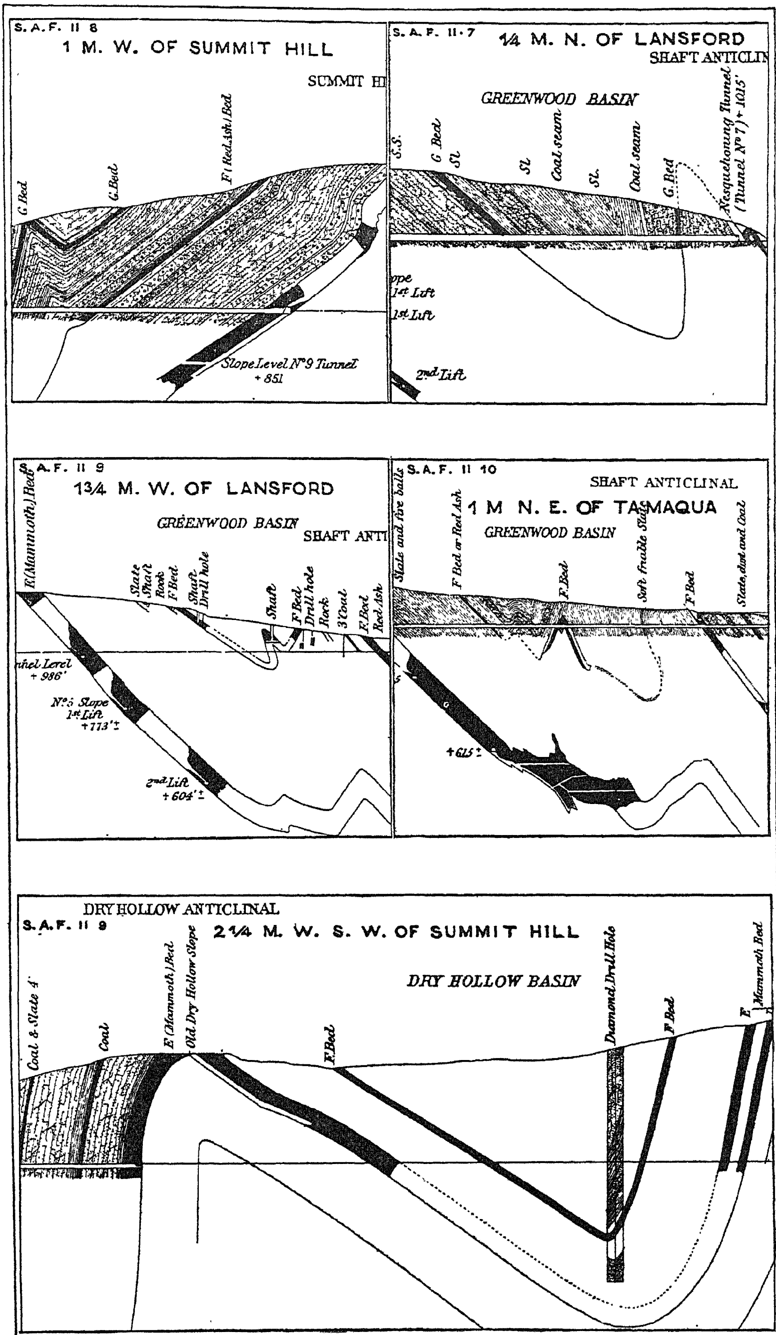
Cross-Sections in the Anthracite-Region.

PLATE XXIX.



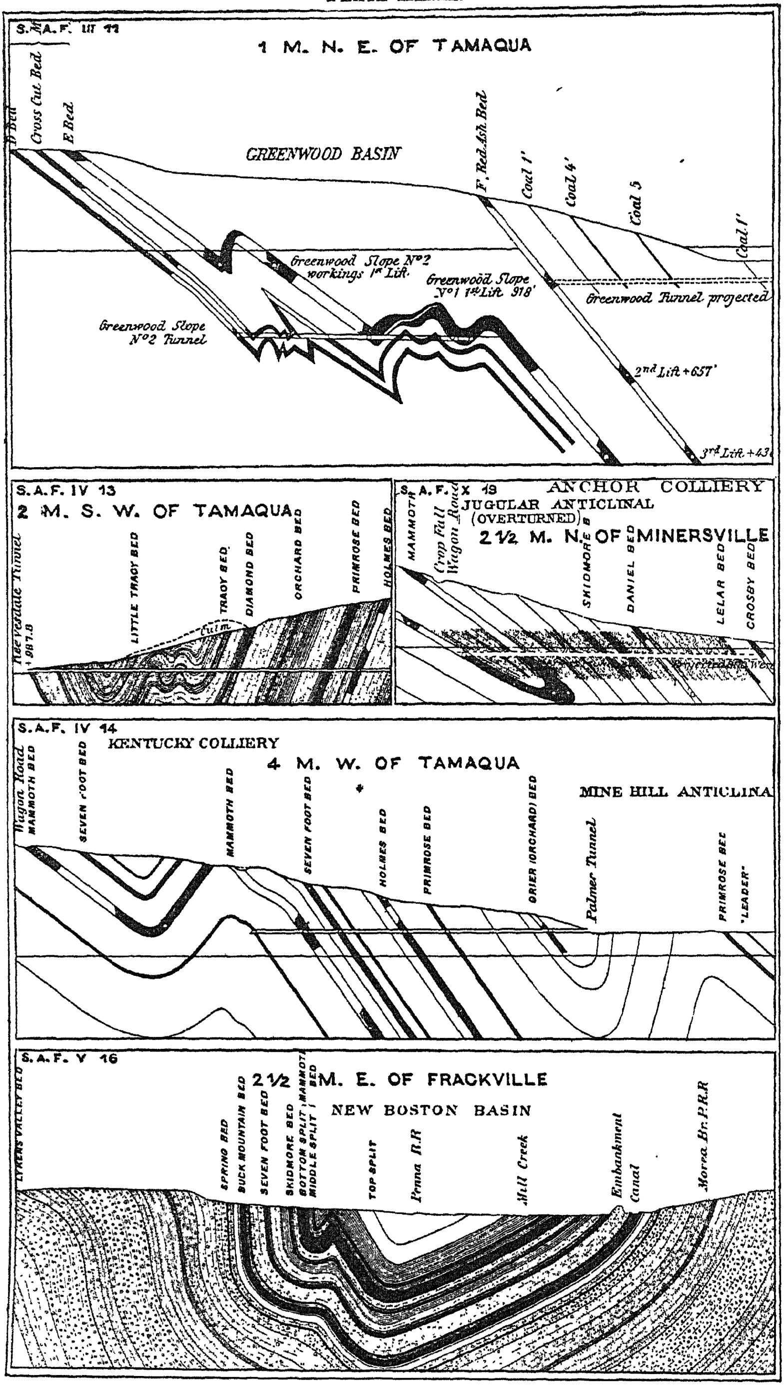
Cross-Sections in the Anthracite-Region.

PLATE XXX.



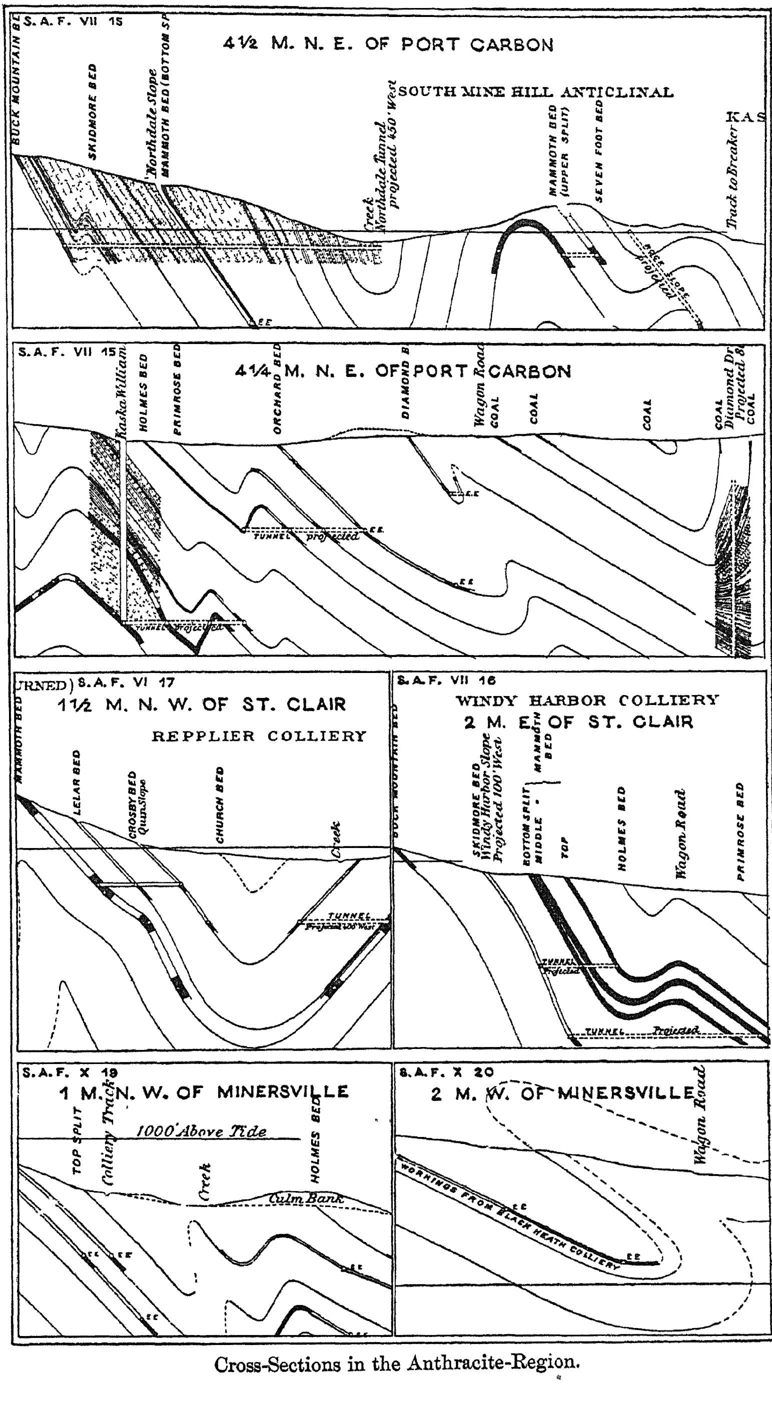
Cross-Sections in the Anthracite Region.

PLATE XXXI.



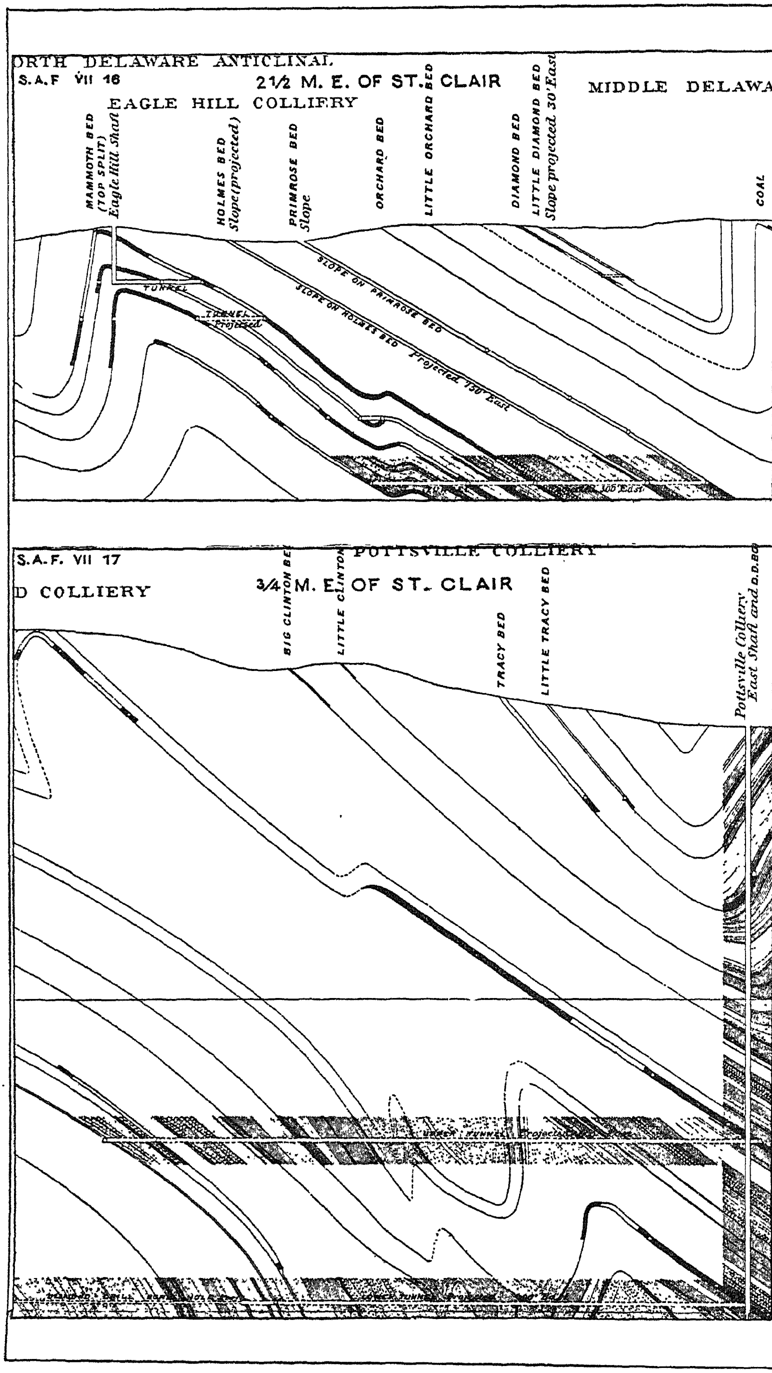
Cross-Sections in the Anthracite-Region.

PLATE XXXII.



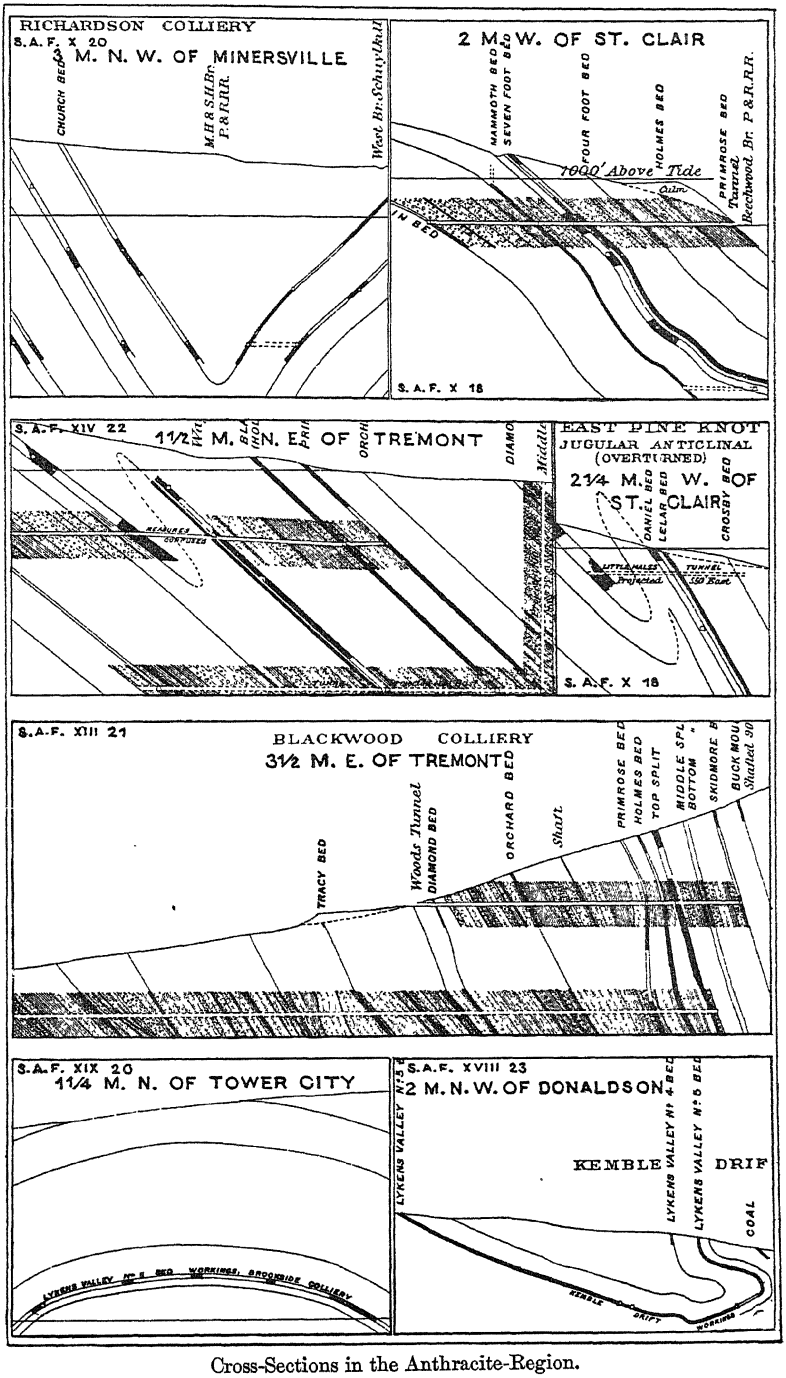
Cross-Sections in the Anthracite-Region.

PLATE XXXIII.



Cross-Sections in the Anthracite-Region.

PLATE XXXIV.



Cross-Sections in the Anthracite-Region.

A Comparison of Recent Phosphorus-Determinations in Steel.

BY GEORGE E. THACKRAY, JOHNSTOWN, PA.

(Atlanta Meeting, October, 1895.)

IN December, 1894, the Cambria Iron Company made a number of heats of Bessemer steel to be used in structures by one of its customers, subject to inspection and tests by a firm of consulting engineers, one of the requirements being that the material should not exceed .08 per cent. of phosphorus.

In other respects the requirements were such that they could only be met by material of good quality; but the general specification is not introduced here, as it is not the purpose of this paper to deal with anything except certain difficulties encountered in determining the phosphorus-content of the material in a manner satisfactory to both seller and buyer.

Preliminary determinations of phosphorus were made in sample-ingots from each of the heats from which material was rolled for the order in question, and these preliminary chemical tests, made in the laboratory of the Cambria Iron Company, showed that, with very few exceptions, every heat was within the prescribed limit of phosphorus. After the material had been rolled, sample-drillings of finished material were taken from the various heats in the regular manner in use at the works, these samples being thoroughly mixed and divided into two parts, one of which was analyzed at the works' laboratory and the other sent to the inspecting engineer.

Several hundred determinations were made from this material without reaching an agreement between the work of the inspector's chemist and that of the works' chemist; but as it is not the object of this memorandum to treat of this part of the subject at length, it is considered sufficient to present herewith only a few of these first determinations. The analyses given in Table I. are those of the original series made on the heats which were afterwards resampled for final determination by an arbitrator; and they are selected for this reason, as the entire comparison is thus rendered more complete.

TABLE I.—*Comparison of Phosphorus-Determinations on First Samples of Finished Material.*

Heat Number.	PER CENT. OF PHOSPHORUS.		Difference.
	Laboratory of Cambria Iron Co.	Laboratory of Buyer's Chemist.	
430,025	.080	.092	+.012
430,026	.074	.088	+.014
430,219	.077	.098	+.021
430,220	.077	.102	+.025
430,230	.077	.110	+.033
430,241	.078	.103	+.025
430,243	.074	.107	+.033
430,277	.074	.100	+.026

The results given in this table show a startling discrepancy. With the exception of heats No. 430,025 and 430,026, the differences between corresponding figures taken from the two reports are very marked; and in consequence of the higher figures obtained by the buyer's representative, as shown by the complete report, the makers were advised that the material could not be accepted. Feeling satisfied, however, that they were right, the makers readily agreed to careful redeterminations of all the heats (about eighty in number) represented in the order. A special inspector was then sent to the works, who personally supervised the taking of new samples—one from every heat—each of which, after drilling, was mixed and divided into three parts.

One part from each sample was analyzed in the works' laboratory, with results which practically corroborated the preliminary and previous analyses made there, while one part of each sample was sent to the buyer's chemist, who made new determinations; but a somewhat similar difference still existed between him and the maker's chemist. The remaining third part of each sample was held in reserve for future use.

In order to close the matter, it was mutually agreed to submit the eight samples which showed the largest differences in the last set of determinations to a certain firm of commercial chemists, by whose results the buyer and seller agreed to be governed. These eight samples were selected from the third portion remaining after the division and distribution of the first two portions of the drillings, the analyses of which,

together with the arbitrator's analyses of the corresponding third portions, are shown in Table II.

In this table the first column contains the results obtained by the buyers' chemist; the second, those of the maker's chemists; and the third, those of the commercial chemists who acted as arbitrators.

TABLE II.—*Comparison of Phosphorus-Determinations on Final Samples.*

Heat Number.	Per cent. of Phosphorus.		
	Buyer's Chemist.	Works' Chemist.	Arbitrator.
430,025	.073	.070	.063
430,026	.090	.077	.075
430,219	.101	.084	.087
430,220	.091	.079	.084
430,230	.083	.075	.077
430,241	.094	.077	.080
430,243	....	.078	.073
430,277	.087	.070	.070

All the results reported by the works' laboratory, shown in the second column of Table II., above, were obtained by the method "S," described hereafter.

Upon receipt of the arbitrator's report, the material in question was accepted by the buyer; but as considerable discussion had taken place regarding the accuracy of various methods for phosphorus-determination, in the hands of different operators and under different conditions, it was decided to enlist the co-operation of various chemists throughout the country, in the hope of throwing some light on this vexed question. For this purpose about forty samples were prepared from each of two open-hearth steel billets, each 5 inches long and 2½ inches square, numbered 19,915 and 19,533 in the tables below. The regular analyses of these two heats made at the works' laboratory are given on page 373.

The usual determinations, made in regular course at the works' laboratory, do not always include arsenic, for the reason that various tests, made from time to time in the past, have shown that, on account of the materials used, little or none of this element is ordinarily present in the finished steel.

TABLE III.—*Ordinary Complete Analysis of Samples Distributed for Phosphorus-Determinations.*

	No. 19,915.	No. 19,533.
	Per cent.	Per cent.
Silicon	.060	.061
Sulphur	.048	.043
Phosphorus	.047	.082
Manganese	.660	.480
Carbon	.410	.280
Arsenic	Slight traces.	

Before sending out these samples, the drillings were taken carefully from all parts of the billet, thoroughly mixed and carefully divided. The samples were sent to a number of steel-works, and to various commercial chemists, by the business department of the Cambria Iron Company, under the immediate direction of Mr. H. H. Weaver, Assistant to the General Manager, whose letter of transmittal advised all that samples had been sent elsewhere, and that tabulated results would be forwarded to each participant when completed.

After the results began to come in, the writer considered that they might be made useful if given to the public; but this was not the intention when the samples were sent out. Before taking any action in the matter of publishing the results, personal letters were written to all the chemists who had kindly assisted in making the determinations, advising them of the proposed publication, and asking for their consent, and for a detailed statement of the method used.

In this first letter to the chemists, relating to publication, they were advised that their names would be considered entirely confidential and referred to only by letters or numbers designating the various chemists and their methods. Further consideration and correspondence led to the conclusion that it would be better to publish the names in full; and, with the consent of all the participants, this is now done, due credit being thus given to all. Under the circumstances, a lengthy discussion by the writer, of the various results and methods, is not deemed advisable, as they are given here in full, so that all those interested may compare them and draw their own conclusions. As compared with a similar series of phosphorus-determinations, which was the subject of a paper presented to

the Institute by Mr. Bachman, in 1882,* it may be safely said, that the present series shows a considerable advance in the art of analytical chemistry; and although the differences shown in Table IV. herewith, might possibly be subjected to criticism on account of certain variations, it could not be said that the differences there shown between the work of different chemists is sufficient to make any serious trouble, if these determinations are to be used as an index of certain qualities of the kind of material under consideration, namely structural steel.

The lowest result in Table IV., on sample 19,915, is .045 and the highest .055 per cent., a total difference of .01 per cent., which, figured on the lowest result as a basis, amounts to 22 per cent. variation.

Similarly, the lowest result on sample 19,533 is .076 and the highest is .091 per cent., a total difference of .015 per cent., or 20 per cent. variation figured on the lowest result as a basis. Although this by no means approaches perfection, and still leaves room for future improvement, it is considered that, on the whole, the results are quite harmonious, and the participants can congratulate themselves thereon with good reason.

Of the differences shown in Table I., the less said the better; but these may be considered to have served a good purpose, in that they were the primary cause of this general investigation, the results of which, in our opinion, should more than restore the confidence in chemical work in general, which was shaken by the set of determinations of which those given in Table I. form a part.

Table IV. shows the work of different chemists on the same samples; and the annexed statements, giving the descriptions of the methods used, are, for the most part, stated in the language of the chemists themselves, as no attempt has been made to edit their descriptions further than to make them harmonious in form.

A careful comparison of the descriptions of the methods used discloses the fact, that several of those given below are practically identical; but, on account of certain small modifications which have been introduced, it is thought best to describe the method of each chemist in full, as given.

* *Trans.*, x., 322.

TABLE IV.—*Comparative Statement of Results Obtained by Different Chemists on Drillings Taken from the Same Piece of Steel (2.5 inches square and 5 inches long) and Thoroughly Mixed Before Division for Distribution.*

	Results of Phosphorus-Determinations.					
	Sample No. 19.915.			Sample No. 19,533.		
	Method.	Per cent. Details. I.	Per cent. Average. II.	Method.	Per cent. Details. I.	Per cent. Average. II.
Andrew A. Blair, Booth, Garret & Blair, Philadelphia, Pa.	A.	.049		A.	.080	
	A′.	.051	.050	A′.	.084	.082
Millard Hunsiker, Engineer of Tests, J. M. Camp, Chemist, Duquesne, Pa., Carnegie Steel Co., Pittsburgh, Pa.	B.	.052	.052	B.	.085	.085
Hugo Carlsson, The Johnson Co., Lorain, Ohio	C.	.050	.050	C.	.080	.080
Townsend V. Church, Chief Chemist, F. Julian, Chemist, Illinois Steel Co., South Chicago, Ill.	D.	.052*	.052*	D.	.088	.088
Albert Ladd Colby, Bethlehem Iron Co., Bethlehem, Pa.	E.	.045	.045	E.	.080	.080
Benedict Crowell, Otis Steel Co., Cleveland, Ohio	F.	.053	.053	F.	.086	.086
Dr. C. B. Dudley, Penna. Railroad Co., Altoona, Pa.	G.	.054		G.	.091	
	G.	.054	.054	G.	.090	.0905
Dr. T. M. Drown, Boston, Mass.	H.	.049	.049	H.	.084	.084
Frank E. Hall, Union Rolling Mill Co., Cleveland, Ohio	I.	.053		I.	.085	
				I.	.087	
	J.	.055	.054	J.	.081	
				J.	.089	.0855
Dr. P. L. Hobbs, Cleveland, Ohio	K.	.046	.046	K.	.080	.080
Fred. D. Jones, Youngstown Steel Co., Youngstown, O.	L.	.052		L.	.086	
	L.	.052	.052	L.	.086	.086
S. W. McKeown, Youngstown, Ohio	M.	.049	.049	M.	.084	
				M.	.085	.0845
R. B. McKeown, Andrews & Hitchcock, Hubbard, Ohio	O.	.049		O.	.082	
	O.	.050		O.	.084	
	M.	.050	.04966	M.	.084	.08333
W. W. McKeown, Cherry Valley Iron Works, Leetonia, Ohio	P.	.049		P.	.083	
	Q.	.049		Q.	.083	.083
	G.	.050	.04933			
A. S. McCreath, Harrisburg, Pa.	N.	.049	.049	N.	.081	.081
W. J. Rattle, Cleveland, O.	R.	.051	.051	R.	.083	.083

* Another determination was made on this sample, but on account of the small size of the portion remaining, the operation and result was unsatisfactory and was so reported at first. For this reason this result is not incorporated in the table.

TABLE IV.—*Continued.*

	Results of Phosphorus-Determinations.					
	Sample No. 19,915.			Sample No. 19,533.		
	Method.	Per cent. Details. I.	Per cent. Average. II.	Method.	Per cent. Details. I.	Per cent. Average. II.
Dr. M. E. Rothberg, Cambria Iron Co., Johnstown, Pa....	S.	.045		S.	.081	
	S.	.046		S.	.082	
	S.	.046		S.	.082	
	T.	.046		T.	.078	
	U.	.046	.0458	U.	.076	.0798
Porter W. Shimer, Easton, Pa.	V.	.047	.047	V.	.081	.081
Frank G. Slocum, Jones & Laughlins, Pittsburgh, Pa..	W.	.047	.047	W.	.080	.080
Oscar Textor, Cleveland, O...	X.	.048	.048	X.	.079	.079
George P. Vanier, Pennsylvania Steel Co., Steelton, Pa.	Y.	.048	.048	Y.	.078	.078
Fred. H. Williams, Riverside Iron Works, Wheeling, W.	Z.	.050		Z.	.090	
Va..................................	Z.	.051	.0505	Z.	.089	.0895
Otto Wuth, Pittsburgh, Pa.....	B′.	.051	.051	B′.	.085	.085
Average........................	....	.0496	.0497	....	.0835	.0832

The averages given above in the first columns, namely, .0496 and .0835 per cent., relating to each sample, are considered to be the correct ones, as in making these, each separate determination is considered, and thus each has equal weight.

If the averages be taken of the mean results of each chemist's work, as shown in the columns marked II., these might be taken to represent the averages of the various laboratories.

In order to make the record as complete as possible, the following notes are given in explanation of the results obtained, and the methods used by different chemists.

Regarding the results reported from his laboratory, Mr. Albert Ladd Colby, of the Bethlehem Iron Works, says that these determinations were made singly in the regular batch of phosphorus-determinations for the day, and run out without any extra care or precaution. He also states that not knowing in the first instance that the results would be published, the matter did not receive his personal attention, nor that of his head chemist, Mr. Buck, as he would have preferred.

Mr. Benedict Crowell, of the Otis Steel Company, reports that he made no determinations for arsenic.

Dr. Charles B. Dudley reports that no special precautions were taken in making the determinations, the whole being done in accordance with his regular routine work.

Mr. Frank E. Hall, of the Union Rolling Mill Co., Cleveland, Ohio, reports that of his determinations on sample No. 19,533, those showing the most divergent results, namely, .081 and .089 per cent., were made by the gravimetric method referred to in the table, and described as the "J" method. His other results on this sample, .085 and .087 per cent., were obtained by Handy's method, referred to and described as the "I" method.

On account of the differences between his determinations, he first concluded that the sample was not uniform, and so reported. At his request, another lot of drillings from the same billet was sent him for further tests, but as the original mixed samples had been exhausted, new drillings were made from all parts of the billet, these drillings being carefully mixed and divided as before. In sending these, his attention was called to the fact that they might not be exactly similar to the original ones, on account of having been drilled subsequently; but as both the first and the second lots were taken from all parts of the billet, it was considered that no very great differences in phosphorus-content should exist by reason of the average samples thus obtained.

With respect to his further investigation on additional samples No. 19,533, Mr. Hall reports as follows:

After having thoroughly mixed the sample, I took out a few grammes for subsequent treatment

The remainder, some 53 grammes, I sifted through a series of sieves.

A 12-mesh sieve retained	. . . . .	6. grammes	$= a$.
A 20- " "	. . . . .	22. "	$= b$.
A 60- " "	. . . . .	19.7 "	$= c$.
A 100- " "	. . . . .	4. "	$= d$.

and but 1.5 grammes passed through the 100-mesh sieve. This amount was so small that although I made a phosphorus test on it, I was unable to satisfactorily finish the determination.

On *a*, duplicate determinations gave	.	.085–.085 per cent. phos.
On *b*, " " "	.	.085–.086 " "
On *c*, one determination "	.	.086 " "
On *d*, " " "	.	.086 " "

I then took that part which I had saved of the original, and made duplicate determinations, giving .085 per cent. and .085 per cent. These were all made by Handy's method.

The above results obtained by Mr. Hall on portions of different degrees of fineness from these sets of drillings, are interesting, showing as they do, a remarkable degree of homogeneity in the samples, which is further evidence tending to show the uniformity of the previous samples, similarly prepared, and confirming the belief that no appreciable lack of uniformity existed on account of the different-sized particles. Previous tests made in the laboratory of the Cambria Iron Company on original samples 19,915 and 19,533, had shown that portions of different degrees of fineness did not vary in phosphorus-content. This was done to make sure that possible errors from this cause were eliminated before sending out these samples.

In further explanation of his results, Mr. R. B. McKeown, of Hubbard, Ohio, reports that when the phosphorus is as low as in sample No. 19,915, he takes two grammes of the sample and doubles the amount of reagents specified in the description of his method, which is herein referred to as method "O."

Samples of the second lot of drillings from billets No. 19,915 and No. 19,533 wese sent to Mr. W. W. McKeown, Jr., who reports that by Dr. Dudley's method he obtained .085 per cent. of phosphorus in sample 19,533; but this result is not incorporated in Table IV. above, for the reason that it was made upon the second set of drillings, as explained heretofore.

The analyses of Dr. Rothberg, made by methods "T" and "U," should not be taken as representative, for the reason that these methods are not regularly practiced in the works' laboratory; and the work done by them should be considered experimental.

With respect to his work, Mr. Fred. H. Williams, of Riverside Iron Works, Wheeling, West Virginia, reports that the method which he used in making the analyses reported upon, is not the one which he uses in every-day work, but he often follows it in special work like the analyses of the sample in question. This method is referred to and described as method "Z."

Arranged in alphabetical order the descriptions of the various methods used by the different chemists in making the determinations shown in Table IV., are as follows:

Method A.

Dissolve 10 grammes of drillings, in a No. 4 Griffin's beaker, in 75 c.c. strong nitric acid. Evaporate the solution to dryness

in the air-bath, replace the cover, and heat until the nitrate of iron is nearly all decomposed. Cool; add 60 c.c. hydrochloric acid; heat gradually until the oxide of iron is dissolved, and evaporate to dryness again in the air-bath. Cool; dissolve in 50 c.c. hydrochloric acid; dilute to about 250 c.c. Heat the solution nearly to boiling, remove the beaker from the heat, and add gradually, from a small beaker, a mixture of 20 c.c. sulphite of ammonia and 40 c.c. ammonia, stirring constantly. The precipitate which forms at first, redissolves, and when all but about 2 or 3 c.c. of the sulphite of ammonia solution has been added, replace the beaker over the heat. If at any time, while adding the sulphite of ammonia solution, the precipitate formed will not redissolve, even after vigorous stirring, add a few drops of hydrochloric acid; when the solution clears, continue very slowly the addition of the sulphite of ammonia. After replacing the beaker over the heat, add to the solution (which should smell quite strongly of sulphurous acid) ammonia, drop by drop, until the solution is quite decolorized, and until, finally, a slight greenish precipitate remains undissolved, even after vigorous stirring. Now add the remaining 2 or 3 c.c. of the sulphite of ammonia solution, which should throw down a white precipitate, which usually redissolves, leaving the solution quite clear and almost perfectly decolorized. Should any precipitate remain undissolved, however, add hydrochloric acid, drop by drop, until the solution clears, when it should smell perceptibly of sulphurous acid. If the reagents are used in exactly the proportions indicated, the reactions will take place as described, and the operations will be readily and quickly carried out. If the solution of sulphite of ammonia is weaker than it should be, of course the ferric chloride will not be reduced, and the solution, at the end of the operation described above, will not be decolorized, and will not smell of sulphurous acid. In this case add more of the sulphite of ammonia, without the addition of ammonia, until the solution smells strongly of sulphurous acid, then add ammonia until the slight permanent precipitate appears, and redissolve in as few drops of hydrochloric acid as possible.

The solution being now very nearly neutral, the iron in the ferrous condition, and an excess of sulphurous acid being present, add to the solution 5 c.c. of hydrochloric acid, to make

it decidedly acid and to insure the complete decomposition of any excess of the sulphite of ammonia which may be present. Boil the solution, while a stream of carbonic acid passes through it, until every trace of sulphurous acid is expelled; then pass a current of sulphuretted hydrogen through it for about fifteen minutes, to precipitate any arsenic which may be present, and, finally, allow the solution to stand in a warm place until the smell of sulphuretted hydrogen has disappeared; or, better, pass a current of carbonic acid through the solution, which will expel the sulphuretted hydrogen in a few minutes. Filter from any precipitated sulphide into a No. 5 beaker; wash with cold water, and to the filtrate add a few drops of bromine-water, and cool it by placing the beaker in cold water. To the cold solution add ammonia from a small beaker very slowly, and, finally, drop by drop, with constant stirring. The green precipitate of ferrous hydrate, which forms at first, is dissolved by stirring, leaving the solution perfectly clear; but subsequently, although the green precipitate dissolves, a whitish one remains, and the next drop of ammonia increases the whitish precipitate or gives it a reddish tint, and, finally, the greenish precipitate remains undissolved, even after vigorous stirring, and another drop of ammonia makes the whole precipitate appear green. If before this occurs the precipitate does not appear decidedly red in color, dissolve the green precipitate by a drop or two of hydrochloric acid, and add a little bromine-water (1 or 2 c.c.); then add ammonia as before, and repeat this until the reddish precipitate is obtained, and then the green coloration, as described above. Dissolve this green precipitate in a very few drops of acetic acid (specific gravity 1.04), when the precipitate remaining will be quite red in color; then add about 1 c.c. of acetic acid, and dilute the solution with boiling water, so that the beaker may be about four-fifths full. Heat to boiling, and when the solution has boiled 1 minute, lower the lamp, filter as rapidly as possible through a 5½-inch filter, and wash once with hot water. The filtrate should run through clear; but in a few minutes it will appear cloudy by the precipitation of the ferric oxide, which has been formed by the exposure of the filtered solution to the air. The points to be observed are the red color of the precipitate, and the clearness of the solution when it first runs through.

Ferric phosphate being white, the red color of the precipitate shows that enough ferric salt was present in the solution to form ferric phosphate with all the phosphoric acid, and enough more to color the ferric phosphate red with the excess of ferric oxide. When the precipitate has drained quite dry, pour about 15 c.c. of hydrochloric acid into the beaker in which the precipitation was made; warm it slightly, so that the acid may condense on the sides and dissolve any adhering oxide; wash off the cover into the beaker; add about 10 c.c. of bromine-water; pour this on the filter containing the precipitate, allowing it to run around the edge of the filter, and let the solution run into a No. 1 Griffin's beaker. Wash out the beaker once or twice, and then wash the filter well with hot water. If the acid in the beaker is not sufficient to dissolve the precipitate completely, drop a little strong acid around the edge of the filter before washing it with hot water.

The scaly film of difficultly-soluble oxide which sometimes forms on boiling the acetate precipitate is caused by the presence of too much acetate of ammonium; but when the instructions given above are carefully carried out, it never appears. Evaporate the solution in the small beaker nearly to dryness to get rid of the excess of hydrochloric acid; add to it a filtered solution of 5 or 10 grammes of citric acid, according to the size of the precipitate of oxide of iron, etc., dissolving in 10 to 20 c.c. of water; then add 5 to 10 c.c. of magnesia-mixture, and enough ammonia to make the solution faintly alkaline. Stand the beaker in cold water, and when the solution is perfectly cold, add to it one-half its volume of strong ammonia and stir it well. When the precipitate of ammonio-magnesian phosphate has begun to form, stop stirring, and allow it to stand in cold water for ten or fifteen minutes, then stir vigorously several times at intervals of a few minutes, and allow it to stand over night. Filter on a small ashless filter, and wash with a mixture of 2 parts of water and 1 part ammonia, containing 2.5 grammes of nitrate of ammonia to 100 c.c. Dry the filter and precipitate, and ignite them at a very low temperature at first, so as to carbonize the filter without decomposing the precipitate, which may then be readily broken up with a platinum wire. Raise the heat gradually, and finally ignite at the highest temperature of the Bunsen burner. When the pre-

cipitate is perfectly white, cool and weigh. Then fill the cruci ble half full of hot water, add from 5 to 20 drops of hydrochloric acid, and heat until the precipitate has dissolved. Filter off on another small ashless filter any silica or oxide of iron that may remain; ignite and weigh. The difference between the two weights is the weight of pyrophosphate of magnesia, which, multiplied by 0.27836, gives the weight of the phosphorus.

Method A′.

This is the molybdate volumetric method (described by Mr. F. A. Emmerton on page 95, *et seq.*, of Blair's *Chemical Analysis of Iron*), with original modifications by Mr. Blair.

Method B.

Five grammes of pig-iron or steel are dissolved in 60 c.c. of nitric acid (1.20 sp. gr.) in a 4.5-inch porcelain dish, with watch-glass cover; evaporated to dryness, still covered; and heated over lamp without cover till all acid fumes are driven off. When cool, dissolve in 30 to 35 c.c. strong hydrochloric acid and evaporate with watch-glass cover till excess of free acid is driven off, as indicated by the first appearance of insoluble ferric chloride on the bottom of the dish. This is dissolved by adding two or three c.c. of strong nitric acid, and the solution is diluted with warm water, filtered, and washed into a 500 c.c. flask. A slight excess of strong ammonia is then added (about 25 c.c.) and then a slight excess of strong nitric acid (about 28 c.c.), until the solution has a clear amber color.

The solution is heated or cooled to 85° C., and 75 c.c. molybdate of ammonia is blown in by means of a pipette; the flask is shaken for five minutes, and allowed to stand till the supernatant liquid is clear. It is then filtered through a weighed filter that has been dried at 100° to 130° C., and weighed between watch-glasses. The precipitate is washed with water containing 2 per cent. of strong nitric acid, dried for one hour, or until the weight is constant, at above temperature, and weighed between watch-glasses. Of the weight, 1.63 per cent. is taken for phosphorus.

Molybdate of Ammonia.—Molybdate of ammonia is made by dissolving 225 grammes molybdic acid in a mixture of 500 c.c.

water and 500 c.c. strong ammonia, and adding this solution to 2500 c.c. of nitric acid of 1.20 specific gravity. This is kept in a warm place over night, and is ready for use.

METHOD B′.

This is the molybdate and magnesia method, modified.

METHOD C.

Dissolve 2 grammes of steel in 75 c.c. nitric acid (sp. gr., 1.13) in an 8-ounce Erlenmeyer flask. Heat on hot plate until dissolved; boil 1 minute. Add 15 c.c. of potassium per manganate solution (5 grammes per 1000 c.c. water) and boil until decomposed. In very high-carbon steel more permanganate may be needed. It must always be in excess, which is shown by a precipitate of manganese dioxide, which remains on boiling. Take the flask from the plate; add a few grains of sugar; boil until clear. Add 13 c.c. of ammonia water (sp. gr., 0.90); shake gently until the hydrate of iron is redissolved; cool the flask to 85° C.; add 50 c.c. of molybdate solution (E. F. Wood's formula of 1888); insert a rubber stopper, wrap in a towel and shake five minutes. Filter through a close filter; wash six times with 1 per cent. nitric acid solution and six times with a solution of potassium nitrate (1 gramme to 1000 c.c. water). Rinse out with clean water the flask in which the precipitation was made; put the filter and precipitate in the flask; add 25 c.c. sodium hydrate solution; shake until the precipitate is dissolved. Add a few drops of phenolphtalein solution, and titrate with standard nitric acid, which has been standardized against steel in which the phosphorus has been carefully determined by a gravimetric method. The sodium hydrate solution is of such strength that 25 c.c. of it equals about 20 c.c. of nitric acid. The standard nitric acid solution is of such strength that 1 c.c. equals .01 per cent. of phosphorus, when 2 grammes of the sample are taken for analysis.

METHOD D.

Dissolve 5 grammes of steel in nitric acid (1.20 sp. gr.); evaporate to dryness and roast thoroughly until no acid odor is perceptible. Take up with hydrochloric acid and add strong nitric

acid. Filter off silica, neutralize with ammonia and make slightly acid with nitric acid. Add 40 to 50 c.c. of molybdate solution and shake five minutes. Allow to stand until the precipitate has settled down well. Filter; dissolve yellow precipitate with citrate of ammonia, and to this add magnesia-mixture. Filter off magnesium pyrophosphate; ignite and weigh.

Method E.

This is Emmerton's well-known method.

Method F.

Dissolve 2 grammes of drillings in a 10-ounce Erlenmeyer flask in 70 c.c. of nitric acid (1.13 sp. gr.); when in complete solution, add an excess of potassium permanganate solution; boil until pink color disappears and brown manganese dioxide separates out; then add ferrous sulphate until the solution is clear; heat to 80° C., and add 50 c.c. of molybdate solution; put into stirring-machine and stir for five minutes; filter immediately, and wash thoroughly with water containing 2 per cent. of sulphuric acid; dissolve the yellow precipitate in as little ammonia as possible; then add 10 c.c. of strong sulphuric acid; dilute with water to 150 c.c.; pass through a Jones reductor, and titrate with standard potassium permanganate solution.

Method G.

Put 1 gramme of the steel in a 10- to 12-ounce Erlenmeyer flask, and add 75 c.c. of nitric acid (1.13 sp. gr.). When solution is complete, boil one minute and then add 10 c.c. of oxidizing potassium permanganate solution. Boil until the pink color disappears and binoxide of manganese separates; remove from the heat and then add crystals of ferrous sulphate, free from phosphorus, with agitation until the solution clears up; adding as little excess as possible. Heat the clear solution to 185° F. and add 75 c.c. of molybdate solution, which is at a temperature of 80° F.; close the flask with a rubber stopper and shake five minutes, keeping the flask so inclosed during the operation that it will lose heat very slowly. Allow it to stand five minutes for the precipitate to settle, and then filter through a 9 c.m. filter and wash with acid ammonium sulphate solution until ammonium sulphide tested with the washings

shows no change of color. Dissolve the yellow phospho-molybdate on the filter in 5 c.c. of ammonia (0.90 sp. gr.), mixed with 25 c.c. of water, allowing the solution to run back into the same flask, and thus dissolve any yellow precipitate adhering to it. Wash until the washings and filtrate amount to 150 c.c.; then add 10 c.c. strong C. P. sulphuric acid, and dilute to 200 c.c. Now pass the liquid through a Jones reductor or its equivalent, wash and dilute to 400 c.c., and then titrate in the reductor-flask with potassium permanganate solution.

* * * * * *

The specific gravities of the reagents given are essential, and the temperature at which the figures are correct is 59° F. In determining these gravities, it is best to use a Westphal balance, but, failing this, a sufficiently delicate hydrometer can be used.

The oxidizing permanganate of potash solution is made as follows: To 2 liters of water add 25 grammes of C. P. crystallized permanganate of potash, and allow to settle before using. Keep in the dark.

The molybdate solution is made as follows: Dissolve 100 grammes of molybdic acid in 400 c.c. of ammonia (sp. gr. 0.96) and filter. Add the filtrate to 1000 c.c. of nitric acid (sp. gr. 1.20). Allow to stand at least twenty-four hours before using.

The acid ammonium sulphate solution is made as follows: To one-half liter of water, add 27½ c.c. of ammonia (0.96 sp. gr.), and then add 24 c.c. strong C. P. sulphuric acid, and make up to 1 liter.

The permanganate of potash solution for titration is made as follows: To 1 liter of water add 2 grammes of crystallized permanganate of potash, and allow to stand in the dark not less than a week before using. Determine the value of this solution in terms of metallic iron. For this purpose, 150 to 200 milligrammes of iron wire or mild steel are dissolved in dilute sulphuric acid (10 c.c. of strong C. P. acid to 40 c.c. of water) in a long-necked flask. After solution is complete, boil five to ten minutes; then dilute to 150 c.c., pass the liquid through the reductor and wash, making the volume up to 200 c.c. Now titrate with the permanganate solution. It is, of course, essen-

tial that the amount of iron in the wire or soft steel should be known. The standard in use in the Pennsylvania Railroad laboratory is a mild steel, in which the iron is known by determining carbon, phosphorus, silicon, sulphur, manganese and copper, and deducting the sum of these from 100 per cent. Not less than two independent determinations should be made, and three are better. The figures showing the value of the permanganate solution in terms of metallic iron should agree to the hundredth of a milligramme in the different determinations.

A very satisfactory method of making and keeping permanganate of potash solution is as follows: Have a large glass bottle holding, say, 8 liters, and two of one-half that size. Paint the outside of these bottles with several coats of black paint or varnish. Fill the large bottle with the standard solution, and after it has stood a proper time, fill one of the smaller bottles from it, without shaking, and standardize. At the same time fill the second small bottle, and refill the large one. When the first small bottle is exhausted, standardize the second one and fill the first from the stock. When this is exhausted, standardize the first again and fill the second from stock, refilling again the stock-bottle, and so on. By this means a constant supply of sufficiently matured permanganate is always available. Of course, if the consumption is very large, larger bottles, or more of them, may be required. Since changes of temperature affect the volume of all solutions, it is desirable that the permanganate solution should be used at the same temperature at which it was standardized. With the strength of solutions above recommended, if the permanganate is used at a temperature differing by 20° F. from that at which it was standardized, the error amounts to less than 0.001 per cent. on a steel containing 0.10 per cent. of phosphorus.

*   *   *   *   *   *

Method H.

Dissolve 2 grammes of the steel in 100 c.c. nitric acid (sp. gr. 1.135) in an Erlenmeyer flask; add 10 c.c. of potassium permanganate (12½ grammes to the liter), and dissolve the brown precipitate by the addition of one-half gramme of ferrous sulphate. Add 100 c.c. of molybdate solution to the solution, and

shake ten minutes, taking care that the temperature never exceeds 50° C. The molybdate solution is made by dissolving 100 grammes of molybdic acid in 400 c.c. ammonia (sp. gr. .96) and pouring this solution into 1000 c.c. nitric acid (sp. gr. 1.2). Wash the yellow precipitate with acid ammonium sulphate; dissolve it in ammonia and precipitate by magnesia-mixture. After complete washing, dissolve the ammonium magnesium phosphate in hydrochloric acid, and into this solution pass sulphuretted hydrogen for three or four hours, the liquid being near boiling the greater part of the time. After filtering off the molybdenum sulphide, and any arsenic which may be present, the ammonium-magnesium phosphate is again precipitated by ammonia, with the addition of a little more magnesia-mixture. The precipitate is weighed as magnesium pyrophosphate.

Method I.

Dissolve 3 grammes of iron or steel in 60 c.c. of nitric acid (1.135 sp. gr.); boil for ten minutes; add 20 c.c. of potassium permanganate solution (15 grammes to the liter); boil two minutes, and add about 0.2 gramme of tartaric acid; boil five minutes. Pig-iron is filtered at this point; steel is not. At a temperature of 90° to 95° C., add 100 c.c. of molybdate solution, made according to Blair's formula; shake ten minutes; filter under pressure; wash with 1 per cent. nitric acid, and afterwards with potassium nitrate solution (1 gramme to the liter); place the filter in the beaker in which the precipitation was made; dissolve in measured volume of standard sodium hydrate solution, add phenolphthalein indicator, and titrate to absence of color with standard nitric acid. Standard solutions are made deci-normal; standardizing against phosphorus by a weighed quantity of yellow precipitate. Carbonates are removed from sodium hydrate by barium chloride solution and filtering.

Method J.

Dissolve 3 grammes of drillings in 50 c.c. of nitric acid (1.42 sp. gr.); evaporate to dryness, and bake on hot plate until residue is perfectly dry and free from acid fumes. Cool; dissolve in as little hydrochloric acid (sp. gr. 1.2) as possible. Evaporate again to dryness and heat until acid fumes are gone. Cool; dissolve in nitric acid (1.42 sp. gr.); boil until hydro-

chloric acid fumes have disappeared; dilute; filter; evaporate filtrate until scum appears on surface of liquid; add 100 c.c. molybdate solution (made according to Blair's formula); mix thoroughly, and allow to stand over night at a temperature of 35° to 40° C.

Siphon off clear solution, filter residue, and wash with molybdate solution of one-half the strength of the above; dissolve precipitate in boiling ammonia (strong), to which one-tenth its volume of hydrochloric acid (1.1 sp. gr.) has been added. After first filterful has run through, add 5 c.c. of magnesia-mixture; then wash filter thoroughly with the hot ammonia wash; stir well, and allow to stand one hour. Filter; wash with ammonia (1 to 2); dry, ignite and weigh.

Method K.

This method is essentially the same as that given in Blair's *Chemical Analysis of Iron*, page 89 *et seq.* The only difference is as follows: Heat the solution to 60°, and then add the molybdate solution, and blow down for thirty minutes. Then add 25 c.c. more molybdate, and blow down for fifteen minutes.

The phosphorus is finally weighed as $Mg_2P_2O_7$; and the details are essentially as Blair gives them.

Method L.

Dissolve 1 gramme of steel in nitric acid (sp. gr. 1.20); when in solution add a few c.c. of hydrochloric acid; evaporate to dryness; burn hard; take up in hydrochloric acid; evaporate until solution is pasty; take up in nitric acid; filter; evaporate to a volume of about 25 c.c.; precipitate at 80° C. with molybdate solution and a few c.c. of ammonia. When thoroughly settled, filter on counterbalanced papers; dry in air-bath forty minutes at 110° C.; cool in desiccator fifteen minutes; weigh; multiply by 1.63.

Method M.

Dissolve from 1 to 2 grammes of drillings in a mixture of equal parts of nitric and hydrochloric acids, using for this purpose a No. V. (8 oz.) Royal Berlin porcelain crucible. Evaporate on a sand-bath, then heat over the flame of a Bunsen burner until the contents of the crucible can be rubbed with a glass

rod to a fine powder, which assumes a dark-bluish purple color. Rest the crucible on a thin iron plate, or wire gauze. From this point the operation proceeds as with an iron-ore, thus: Dissolve in hydrochloric acid; evaporate to a pasty consistency; take up finally with nitric acid; dilute; filter into a flask, heat to about 180° F., add 30 c.c. molybdate solution, and shake gently for about one minute. If the precipitate does not appear promptly, add a few drops of strong ammonia and continue the agitation for a few seconds longer. Weigh on a counterbalanced filter, after drying in a hot-water oven until a blue stain appears on the filter-paper containing the precipitate. If the filter shows a brown residue after filtering the nitric acid solution, it is considered that it is not burned sufficiently to destroy the combined carbon before redissolving in hydrochloric acid. In such cases, low results are likely to be obtained. With very low phosphorus, the weight of the yellow precipitate is taken at the end of about three minutes after taking the filters from the oven. In the case of high phosphorus, the filter and contained precipitate are cooled in a desiccator before weighing.

Method N.

This is the acetate method. Dissolve 5 grammes of the metal in nitric acid with only a few drops of hydrochloric acid mixed with it to aid solution. Evaporate nearly dry, then add hydrochloric acid and evaporate to complete dryness. Redissolve in hydrochloric acid, dilute and filter. Reduce with bisulphite of ammonia, taking care not to add an excess. Separate arsenic with hydrogen sulphide, and filter off, after allowing it to stand aside for some time. Boil off excess of hydrogen sulphide; add a small crystal of chlorate of potash to oxidize a little iron. Neutralize with ammonia until a permanent precipitate is formed, then precipitate with acetate of soda. Boil well and filter. Dissolve the precipitate in hydrochloric acid; evaporate down; add citric acid and ammonia, and filter if necessary. Precipitate the phosphorus with magnesia-mixture; add a little more ammonia and stir thoroughly; allow it to stand not less than six hours; then filter, ignite and weigh as usual.

Dissolve the precipitate in a little dilute hydrochloric acid, and if any residue remains, filter off and deduct from the original

weight. The main points are: not to have any sulphurous acid present; to avoid too large a precipitate with the acetate of soda; and to avoid too great an excess of citric acid. Use considerable excess of magnesia-mixture. Mr. McCreath also says, with regard to the separation of the arsenic with hydrogen sulphide: "without this separation my results would have shown .053 per cent. and .086 per cent. of phosphorus in the two samples."

Method O.

Dissolve 1 gramme of steel in 35 c.c. of nitric acid (sp. gr., 1.13). Add a few crystals of permanganate of potash, boil for two or three minutes; reduce the permanganate with nitrite of soda. Add 5 c.c. of ammonia (sp. gr., 0.90); cool to 85° C.; add 30 c.c. of molybdate solution. Shake for about three minutes and put in warm place until it settles clear, which takes generally from five to ten minutes; filter, wash with water containing 2 per cent. of nitric acid, then with an acid solution of sulphate of ammonia. Dissolve the precipitate in ammonia; reduce with zinc, and titrate with permanganate.

Method P.

Dissolve 1 gramme of steel in 15 c.c. of hydrochloric and nitric acid (1 to 1); evaporate to dryness; heat one-half hour on gauze over Bunsen burner. Dissolve in hydrochloric acid; evaporate to pasty mass; add 15 c.c. of nitric acid (1.20 sp. gr.); heat a few minutes. Filter, wash and precipitate by 30 c.c. of molybdate solution made by Wood's formula; the temperature of the solution after addition of molybdate solution being 40° C. Filter through counter-balanced filters; wash precipitate with 2 per cent. nitric acid. Dry in water-oven and weigh from desiccator. Factor used, 1.63 per cent.

Method Q.

Dissolve 1 gramme of steel in 21 c.c. of nitric acid (sp. gr., 1.13); heat half an hour, or until carbon is dissolved, at 75° to 90° C. Oxidize by saturated solution of permanganate of potash. Keep solution well colored for thirty minutes at 75° to 90° C. Reduce manganese dioxide by potassium nitrite solution; filter and precipitate with 30 c.c. of molybdate solution. The yellow precipitate is weighed as in method P.

Potassium nitrite is used, as it adds only such compounds as are already present, and avoids the presence of sulphuric acid. Solution in nitric acid, if not heated over 90° C., gives silica in such condition that, with the above amount of acid, pig with 2 per cent. of silicon can be carried through without precipitation of silica or silico-molybdate of ammonia.

METHOD R.

This is the standard method given in Fresenius's *Quantitative Analysis.*

METHOD S.

Take 1 to 2 grammes of steel, according as the phosphorus is high or low; put in a 12-ounce beaker; add 15 to 25 c.c. of nitric acid (1.13 sp. gr.); as soon as violent action ceases, put on the lamp; bring to a boil; oxidize with permanganate; dissolve the precipitated manganese dioxide with sugar or ferrous sulphate; boil one minute longer; cool to about 40° C., add 50 to 60 c.c. of molybdate solution; shake for five minutes; then filter, using a 9 c.m. Schleicher and Schuell, No. 590 filter; rinse out the flask with 1 per cent. nitric acid, and wash four times on the filter with the same; then wash five times with potassium nitrate (3 grammes per liter) to remove free nitric acid.

Transfer paper and precipitate to a 5-ounce beaker; add from a burette enough standard soda to dissolve the precipitate; dilute to 50 c.c. with water; add two drops of phenolphthalein, and titrate at once with a standard solution of nitric acid.

As the compound formed by dissolving the yellow precipitate in caustic soda is very volatile, it is essential that the titration be made promptly.

If the solutions are of such strength that 1 c.c. of alkali equals 1 c.c. of acid, and 1 gramme of steel is taken for a determination, then the combined alkali in c.c. $\times .02 =$ per cent. phosphorus (it is assumed that $\frac{n}{6}$ solutions are used).

METHOD T.

This is the acetate method as given on page 81 *et seq.*, of Blair's *Chemical Analysis of Iron.*

Method U.

This is Wood's chromic acid method, with direct weighing of yellow precipitate, as given on pages 102 and 103 of Blair's *Chemical Analysis of Iron.*

Method V.

Dissolve 5 grammes of steel in nitric acid (sp. gr. 1.20); evaporate to hard dryness on hot plate; dissolve in as little hydrochloric acid as possible, and again evaporate to hard dryness on hot plate. The reason for this double evaporation to dryness is, that a simple baking with nitric acid alone is not enough to insure complete oxidation of the phosphorus in some samples. The second baking with hydrochloric acid insures full oxidation. Now redissolve in hydrochloric acid and filter from silica. Replace hydrochloric acid with nitric acid by evaporation, and precipitate with ammonium molybdate solution (100 grammes molybdic acid, 400 c.c. ammonia of 0.96 sp. gr., and 1000 c.c. of nitric acid of 1.20 sp. gr.) at about 50° C. Filter, and wash well with acid ammonium sulphate solution. Dissolve in not too great excess of ammonia, allowing the ammoniacal solution to run into the beaker in which the precipitation was made. In case this solution is not perfectly clear, or in case any insoluble matter remains upon the filter, add a few drops of hydrochloric acid (1.12 sp. gr.), and allow the acid solution to run into the ammoniacal solution, until the latter becomes slightly acid, throwing down the yellow precipitate. Boil for a few minutes. This boiling seems to decompose certain phosphatic compounds. Now add ammonia in slight excess and boil a few minutes. Filter and wash with hot water. Dissolve the precipitate in a few drops of nitric acid, and precipitate with ammonium molybdate solution. Filter, wash, and dissolve in ammonia the additional yellow precipitate, and add it to the main solution. Add a little hydrochloric acid to the ammoniacal solution, but not enough to render it acid. Precipitate by magnesia-mixture, with stirring. Filter, and wash a few times with dilute ammonia (1 part ammonia of 0.96 sp. gr. to 3 parts of water). Redissolve in hydrochloric acid, and let the solution run into the beaker in which the precipitation was made. Add sulphurous-acid water, and boil until

excess of sulphurous acid is driven off. Now add strong sulphuretted-hydrogen water and heat gently until all arsenic and molybdenum are separated as sulphides. Filter and reprecipitate by adding excess of ammonia, and a few cubic centimeters more of magnesia-mixture. Filter and wash with dilute ammonia, and weigh as magnesium pyrophosphate.

Method W.

This is the chromic acid method of E. F. Wood, published in the *Journal of Analytical Chemistry*, vol. i., part 2, April, 1887.

Weigh 1.63 grammes steel into a 6-ounce Erlenmeyer flask, and add 35 c.c. of nitric acid (sp. gr., 1.20). When violent action has ceased, place over lamp; boil until carbonaceous matter is dissolved and solution is concentrated to 15 c.c.; add 15 to 20 c.c. of chromic acid solution; boil down to 15 c.c.; add 5 c.c. of water; cool; add 50 c.c. of molybdic acid solution; shake well; let stand at temperature of 20 to 30° C. until yellow precipitate is completely settled and solution is perfectly clear.

Prepare a weighed filter by drying a 7 c.m. filter of Munktell's No. 1 Swedish paper for 35 minutes in steam-drying bath at 100° C.; weigh rapidly to within 1 milligramme.

Remove most of the supernatant liquid with a siphon, filter remainder and wash precipitate on to filter with water containing 2 per cent. of nitric acid (sp. gr., 1.20); wash precipitate and filter four or five times with this mixture; dry at 100° C. for 45 minutes (the point of filter should be blue when dried enough); weigh.

The yellow phospho-molybdate of ammonia contains 1.63 per cent. of phosphorus, so that each milligramme equals .001 per cent. of phosphorus in the sample.

Reagents.

Molybdic Acid Solution.—To a mixture of 1200 c.c. of water and 700 c.c. of concentrated ammonia (sp. gr., 0.88 to 0.90) add 1 pound of molybdic acid; stir till dissolved; add 300 c.c. of nitric acid (sp. gr., 1.42); cool; pour 700 c.c. of this solution into a mixture of 630 c.c. of nitric acid (sp. gr., 1.42) and 1600 c.c. of water; let stand 24 hours, and filter as desired for use.

Chromic Acid Solution.—Dissolve about 50 grammes of chromic acid in 1 liter of nitric acid (sp. gr., 1.42) by warming.

Method X.

This is the same as Method W, with the following precautions: Heat the final solution for 10 minutes at 65° C., either agitating during that period and filtering at once, or, if the solution is not stirred or agitated, allowing it to stand over night. The filter-paper is weighed at the end of one minute from the time of taking it from the air-bath, and the same time is allowed before taking the weight of the filter and precipitate. A correction for a constant uniform *plus* error is made by deducting .001 (one-thousandth) for percentages from .01 to .03 of phosphorus and .002 (two-thousandths) for percentages from .03 to .07 and .003 (three-thousandths) for percentages from .07 to .15.

Method Y.

This is the "alkali" method, using potassium permanganate for oxidizing and sugar for taking up manganese dioxide. Wood's formula is used for molybdate, and titration is made with nitric acid.

Method Z.

Dissolve 3 grammes of the drillings in 36 to 40 c.c. nitric acid (1.20 sp. gr.) in a 6-inch evaporating dish, covered; evaporate to hard dryness; redissolve in strong hydrochloric acid; evaporate to remove excess of acid; add 25 or 30 c.c. of strong nitric acid and evaporate to small bulk, but avoid going low enough to form insoluble oxide of iron on the side of the dish. Dilute with water, filter off silica and wash. Add ammonia carefully, with stirring, until solution is amber-tinted, avoiding very dark tint. Add 75 c.c. of molybdate solution (E. F. Wood's formula). Stir and place on hot plate, where temperature gradually rises to about 50° C. It is then placed where it will gradually fall to ordinary temperature, and allowed to stand over night. The next day it is filtered, washed with acid ammonium nitrate solution (1200 c.c. of water, 500 c.c. of nitric acid of 1.20 sp. gr. and 400 c.c. ammonia). Dissolve through filter with ammonia; precipitate with magnesia-mixture and

weigh as pyrophosphate, without any final treatment of precipitate to determine its purity.

In conclusion, it is desired to give credit and thanks to all whose names appear above, and all who assisted in the determinations. The writer does not claim any originality, as he has merely made a compilation—not, however, without considerable work and correspondence—of the methods used and the results obtained by many others. It is hoped that the showing made herein with respect to phosphorus determinations will be of some value; and it is the writer's opinion that it should carry considerable weight as evidence of the general accuracy of such work, and may be considered good testimony as against the pessimistic opinions so often encountered in discussions of this subject.

A few samples of the second lot of drillings from heats Nos. 19,915 and 19,533 are still in the possession of the writer, who will be glad to send small portions thereof to any one interested in checking results with those shown in Table IV.

The Determination of Graphite in Pig-Iron.

BY P. W. SHIMER, EASTON, PA.

(Atlanta Meeting, October, 1895.)

THE purpose of this note is to call attention to a source of error in the determination of graphitic carbon, made by the usual method of solution in hydrochloric acid. Although the method is tedious, because of the necessary treatment of the separated carbon with caustic potassa, alcohol and ether, the text-books seem to give it preference; and it is, perhaps, used more generally than the method of solution in dilute nitric acid. Solution in hydrochloric acid usually gives higher graphitic-carbon results than solution in nitric acid, and many, therefore, consider it more trustworthy, the inference being that the lower results obtained by nitric acid are due to the loss of some of the finely-divided graphite by reason of the oxi-

dizing action of the solvent. But, experiments made in Dr. Drown's laboratory, about seventeen years ago, showed no appreciable oxidation of graphite in the fifteen or twenty minutes' boiling required for the solution of a sample of pig-iron.

The point I desire to bring out here is, that the high results in graphitic carbon obtained by solution in hydrochloric acid are due to the presence, in the graphitic residue, of titanium carbide (see *Trans.* xv., 455), and possibly of other insoluble carbides, the carbon of which is, of course, included with the graphite in the final determination. In the nitric acid method, the titanium carbide is easily dissolved, and its carbon appears with the combined carbon, when the latter is determined by difference between graphitic and total carbon.

The method by solution in dilute sulphuric acid is open to the same objection as that by solution in hydrochloric acid; for titanium carbide is insoluble in sulphuric acid, and, I may add, it is also unattacked by hydrofluoric acid, and by a boiling solution of caustic potassa.

The following is an analysis of a pig-iron unusually high in titanium:

	Per cent.
Silicon,	3.650
Phosphorus,	1.145
Sulphur,	0.010
Manganese,	0.226
Graphitic carbon,	3.206
Combined carbon,	0.128
Titanium,	0.399
Iron (by difference),	91.236
	100.000

The total carbon in this iron was determined by dissolving in an acidified solution of double chloride of copper and potassium, and subsequent combustion. The graphite was determined by solution in dilute nitric acid and combustion. A determination of graphite, made by solution in hydrochloric acid and combustion, gave 3.327 per cent. of graphite, a result 0.121 per cent. higher than that obtained by the nitric acid method. The amount of carbon combined as titanium carbide (TiC) with 0.399 per cent. of titanium is 0.1 per cent., which counts as graphite in the determination by solution in hydrochloric acid. The results may be set down as follows:

	Per cent.
Total carbon,	3.334
Graphite by nitric acid solution,	3.206
Graphite by hydrochloric acid solution,	3.327

The error in the hydrochloric acid method falls heavily upon the resultant estimate of combined carbon, which is determined by difference, as appears below:

	Per cent.
Combined carbon, when graphite is determined by nitric acid,	0.128
" " " " hydrochloric "	0.007

An experiment was made to determine the action of boiling nitric acid (1.20 sp. gr.) on the graphite from this iron. A sample of two grammes was dissolved in hydrochloric acid (1.10 sp. gr.). After washing the graphitic residue with water, it was boiled gently for one hour with nitric acid (1.20 sp. gr.), with the addition of a little water from time to time, to keep up the bulk of the solution. The graphite, as thus determined, was 3.203 per cent. against 3.206 per cent., by direct solution in nitric acid, showing that the treatment, for one hour, with boiling nitric acid, had dissolved out the titanium carbide without having attacked the graphite. The graphite in this high-silicon iron, however, was coarse and perhaps unusually resistent to the oxidizing action of nitric acid. It is proposed to make similar experiments on the graphite from a variety of pig-irons.

The 0.128 per cent. of combined carbon is made up as follows:

	Per cent.
Carbon combined with 0.399 per cent. titanium as TiC,	0.100
Combined carbon soluble in hydrochloric acid (probably combined with iron and manganese),	0.007
Carbon possibly existing as insoluble carbide other than titanium carbide,	0.021

A careful mechanical separation of a few grammes of titanium carbide was made from several pounds of this iron by use of the long, slightly inclined glass plane described in the paper before the Institute referred to above.

Besides titanium and carbon in this separation, there is some vanadium, apparently also existing as an insoluble carbide, which would account for a part of the above 0.021 per cent. of

combined carbon. This investigation is, however, still under way.

The writer has never encountered a pig-iron free from titanium; the amount found varying usually from 0.05 to 0.40 per cent. In irons with a coarsely crystalline fracture, the cubical crystals of titanium carbide may always be found when carefully looked for. The conclusion seems to be fair that the hydrochloric acid method includes, with the graphite determined by it, the carbon existing as insoluble carbide of titanium. With pig-irons containing from 0.05 to 0.40 per cent. of titanium, the graphite so determined will be from 0.013 to 0.100 per cent. too high, while the combined carbon will be correspondingly low.

It follows that more light would be thrown upon the condition of the carbon in pig-iron by making three determinations, viz.: one of total carbon; one of the carbon insoluble in hydrochloric acid; and one of graphite by the nitric acid method. We would thus have determinations of graphitic carbon; carbon combined with iron and manganese, soluble in hydrochloric acid; and carbon combined as carbides insoluble in hydrochloric acid. In high-silicon, low-sulphur titanic irons, the insoluble form of combined carbon exceeds the carbon existing as soluble carbides of iron and manganese. It is important to know how the carbon is combined. One-tenth per cent. of carbon, combined with titanium in the condition of disseminated microscopic crystals, probably has no effect on the hardness of pig-iron; while the same amount of carbon, combined with iron and manganese, would have an appreciable hardening-effect. Practically, therefore, it may be desirable to have the carbon existing as carbides insoluble in hydrochloric acid appear with the graphite as determined by the hydrochloric acid method, although the actual graphite can be determined only by solution in nitric acid. At all events it is essential to know by what method graphite has been determined, in order to draw conclusions from determinations of graphitic and combined carbon in pig-iron.

Notes on the Magnetization and Concentration of Iron-Ore.

BY WILLIAM B. PHILLIPS, BIRMINGHAM, ALA.

(Atlanta Meeting, October, 1895.)

THE concentration of natural magnetites has been carried on in this country for several years, and more or less information has been collected on the subject. Various inventors, availing themselves of the magnetism which is inherent in such ores, have devised machines for separating the impurities from the ore by means of the electric current; and a greater or less degree of success has rewarded their painstaking and persevering efforts. We are now fairly in position to judge of the value of the process as applied to such material. It is not my purpose to speak of this matter further than may be required by the emergencies of the case. I shall assume that most engineers and metallurgical chemists are already acquainted with what has been done at the Chateaugay mines, at Mineville, at the Benson mines, at the Weldon and Ogdensburgh mines, and at Cranberry; although, at this latter place, there has not been so much done as we could wish. I shall not now speak of the various types of separators. From the time of William Fullarton, to whom was granted an English patent, in 1792, for separating iron-ore by the application of magnetic attraction, down to the present time, a great many devices have been suggested, and a great many patents taken out for this purpose; but it is not necessary now to enter upon a discussion of their merits. Perhaps, at some future meeting of the Institute, after we shall have experimented further with some of the machines on the market, there may be an opportunity to contribute something to the current knowledge of the efficiency and economy of separating-machines.

In this paper I propose to give, as briefly as possible, the results already reached in converting a non-magnetic ore into magnetic ore, and then concentrating it over a magnetic separator of the alternate-polarity type. I am aware of the work

that has been done by Schneider et Cie., in Savoy, on the carbonate ores of the Allevard district; and I published in the *Engineering and Mining Journal* (vol. lviii., p. 200) a translation of an article on the subject by Lürmann that appeared in *Stahl und Eisen*, July 15, 1894. The magnetization of the franklinite ores, at Bethlehem, and the separation of the zinc-bearing compounds from certain brown ores of southwest Virginia, have, also, not been forgotten. But the process I am about to describe differs radically from any of these, as I think will be apparent as the matter is unfolded.

During the late winter of 1893, Mr. George B. McCormack, at that time Assistant General Manager of the Tennessee Coal, Iron, and Railroad Company, Birmingham, Ala., noticed that a piece of soft, red fossiliferous ore, that had fallen into a coke-oven flue at Pratt mines, and had been subjected, at a full red-heat, to the action of the reducing gases filling the flue, was quite black and resembled magnetite. A trial with a hand-magnet showed that it was magnetic; and after consulting with Mr. Alfred E. Barton, Superintendent of the Ensley furnaces, they concluded to try whether or no such ore could be made magnetic uniformly under the same conditions as produced magnetism in the small piece. Accordingly, they placed several lumps in the flue, and after a while observed that they became highly magnetic, and were capable of being concentrated; that is, on pulverizing, the magnetic portion was easily removed from the sandy portions, and it was possible, in this way, to increase the metallic iron in the ore from 47 per cent. to over 60 per cent. The early analyses were made by Mr. J. H. Pratt, who at that time was conducting an analytical laboratory in Birmingham. After satisfying themselves that the ore could be magnetized in this way, Messrs. McCormack and Barton secured patents on the process of magnetizing hematite ores and then concentrating them magnetically.

It has been known, for many years, that non-magnetic iron-ore could be rendered magnetic by passing over it, at red-heat, some reducing gas, and this principle has been recognized for the detection of iron-ore before the blow-pipe for at least forty years. There is nothing new in the statement of the fact that a reducing gas, at a sufficient temperature, renders non-magnetic ore magnetic; but the proposal to make use of this prin-

ciple to the commercial concentration of a particular class of ore comes well within the domain of useful projects. It is well known to those who have had to do with heap- and kiln-roasting of brown ore, that it becomes magnetic; and Mr. Clemens Jones has proposed* that such reactions may be utilized in the concentration of this kind of ore. In such operations the partial reduction of the ferric oxide to ferrous oxide confers upon the ore sufficient magnetism to permit its treatment on a magnetic separator with reasonable hope of increasing the content of iron and decreasing the content of siliceous matter, thus making the ore a better material for use in the blast-furnace.

When Messrs. McCormack and Barton proposed to avail themselves of the partial conversion of ferric oxide into ferrous oxide, with the formation of magnetic oxide, they did not think to announce a new principle in chemistry. Had they stopped here, there would have been nothing in the idea but the restatement of a fact well recognized for a long time. But they proposed to go a step farther, and to concentrate this partially-reduced ore over a magnetic-separator, and to apply the idea to a certain kind of ore, viz., hematite ore. They availed themselves of the most convenient means at hand, and after finding that the ore under examination became magnetic, and that the sandy portions of it could be separated from the more ferruginous portions by a magnet, they based upon this a scheme for the betterment of the ore.

I took charge of the matter for the Tennessee Coal, Iron and Railroad Company in the spring of 1893, and since that time have done a great deal of work on it, both in the laboratory and in an experimental plant erected at Ensley, Ala., treating 3000 pounds of ore at a time; and at Bessemer, Ala., where, for some weeks, we magnetized ore in a Davis-Colby gas-fired kiln holding 110 tons.

In the conduct of the laboratory-experiments and the carrying out of the analyses necessary for the elucidation of the subject, I have been most kindly assisted by Mr. P. H. Haskell for a few months at Ensley, and by Mr. J. R. Harris, Assistant Chemist in the Birmingham Laboratory of the Tennessee Coal,

* *Trans.*, xix., 289.

Iron and Railroad Company. To both these gentlemen I desire to express my acknowledgments of their services, especially to Mr. Harris, who has done a great deal of valuable work since Mr. Haskell was transferred to the engineering corps. That more has not been done is to be explained by the fact that all the usual analytical and technical work of the company has been carried on at the same time and without a break, amounting during the last 8 months, when we have been especially concerned with this matter, to 2000 analyses. There has been no opportunity to work exclusively on the magnetization and concentration question. Still, we have done something; a great many analyses and calculations bearing on the subject have been made, and we feel that the time has come when we may communicate to the profession a part of what has been accomplished, reserving the remainder for a more convenient season and for more accurate information. The results that have been reached are based partly on the small plant at Ensley, now dismantled, and partly on the larger plant at Bessemer, where we made some 40 tons of concentrates before changing the kiln to hard and brown ore. After satisfying ourselves as to some of the most important points connected with the process, and concentrating ore from 44 per cent. of iron to 61 per cent., the emergencies of the iron-trade at the time compelled us to forego further work in this direction and to use the kiln for calcining brown ore. During the meantime, however, we are preparing to enter upon the business of magnetizing and concentrating the lower grades of our iron-ores upon a much larger scale than heretofore. The plant is being overhauled and enlarged, modifications of the kiln are in progress, and a new separator is being constructed, especially designed for this particular business.

I will now proceed to describe briefly what has been done, and the manner of conducting the operations involved in the process.

The deposit of red fossiliferous ore (Clinton) attains its maximum thickness in the immediate vicinity of Birmingham, where the Eureka seam (now termed Ishkooda) is from 18 to 24 feet thick. The upper portion of this seam, near the outcrop, is what we term soft ore, inasmuch as the lime has been removed by leaching. Under cover the ore becomes hard and the

amount of lime it carries varies from 12 to 25 per cent. In mining the soft ore it is customary to remove the over-burden and to take the ore from open cut, the tracks being at different levels to facilitate the handling. The over-burden varies in thickness from a few feet to 40 feet, and is stripped on the dip for distances varying from 50 to 300 feet from the crop. The dip of the seam increases as one goes towards the southwest, the average being close to 20°. In the early years of iron-making in the district it was customary to remove from 15 to 20 feet of the ore and to send it all to the furnace, but of late the mining has been restricted to 10 or 12 feet and there has been left in the ground from 8 to 10 feet of ore. This lower portion of the seam is now considered too low in iron and too high in silica to permit its profitable use in the furnace. It carries about 40 per cent. of iron and about 35 per cent. of silica, the silica increasing with the vertical depth below the mining mark. Not less than 500,000 tons of this low-grade ore is now stripped, the upper 10 or 12 feet of workable ore having been removed and sent to the furnaces. Nothing remains now but to shift the tracks and to mine the lower portion also, thus making the entire thickness of the seam available for the furnace. With the exception of the Irondale seam, 5 feet thick and carrying about 53 per cent. of iron, at no point on the mountain can the entire seam be mined for furnace-purposes unless the lower portions be subjected to some process of concentration, The Irondale seam is distinct from the big, or Ishkooda, seam; it carries from 6 to 8 per cent. more of iron, also more alumina, and can be profitably mined from wall to wall. This, however, is not the case with the Ishkooda seam. It is not likely that, on the average, more than one-half of it can be used now for the manufacture of iron, and, unless the remaining portion can be concentrated, it is practically of no use whatever. The stripping that has been done is chargeable to the ore mined and sold, so that the lower portion of the seam can be mined at a very slight expense, probably for about one-half of the expense now incurred. It can be mined and put on the car for 20 cents per ton, and can be put into a kiln on the mountain for 25 cents a ton. These are not haphazard figures, but are based on what we know to be well within the actual cost. For purposes of calculation, however, I prefer to

take the higher figure, and to say that this lower portion of the Ishkooda seam can be put into the magnetizing kiln on the mountain for 25 cents per ton and leave a fair margin of profit to the miner. It can be loaded into the railroad-cars for the same sum, and can be laid down in any stock-house in the vicinity of Birmingham for 40 cents per ton. This statement applies to such ore as has been already stripped, and from which the upper portion has been removed for use in the furnace, leaving the question of tracks and loading-appliances already provided for. It applies, therefore, to what may be considered a limited amount of ore, and it is so in a certain sense and as compared with the enormous deposit of such low-grade material along the mountain. A concentrating-plant taking 500 tons of ore per day and working steadily 365 days in the year would require nearly 3 years to use up what is now ready for mining; and when we consider that the mining of the upper portion is going on all the while, thus increasing the amount of the low-grade ore left, it is not likely that such a plant would be able to use the uncovered portion in five years, if the removal of the better ore should cease on the first day of October, 1895. So much for the Ishkooda opening; but there are several other mines along the mountain, within 2 and 3 miles of Ishkooda, that exhibit the same conditions, and I feel warranted in saying that the available supply of 25-cent ore will not fall much short of 1,000,000 tons.

When one considers the immense amount of low-grade ore that has not been touched, from Grace's Gap to Gate City, a distance of 9 miles, one can not fail to be satisfied that for many years to come the supply of ore for a concentrating-plant will be ample. With a full knowledge of the subject and from an acquaintance of several years with the ore-situation in the Birmingham district, I have no hesitation in saying that a concentrating-plant of the capacity mentioned above, would not experience any serious difficulty in obtaining low-grade ores suitable for concentration and at a price that would leave a fair profit, for twenty years.

These statements are made at this time, not by way of digression but to show why we entered upon the matter at all. The supply of cheap ore is the most important factor in the whole enterprise. Had we not been satisfied as to this, we

would not have taken hold of the matter. The price at which the ore can be delivered to the magnetizing-kiln was the first question to be settled, for upon this depends the profitableness of the undertaking. This will be more apparent when we come to the discussion of the inevitable loss in conducting the process: loss in moisture driven out, in handling, and in the tailings from the separator. It is the key to the whole situation. It alters the conceptions we have hitherto had as to the conditions that must prevail in the successful magnetic concentration of iron-ore, and allows us to carry a percentage of iron in the tailings that would signify ruin to one whose notions of efficient work had been derived from observations of plants now engaged in the concentration of natural magnetite. As the cost of such operations is chargeable against the first cost of the ore, in respect of the amount of ore required for one ton of concentrates, it follows that the cheaper the ore, other things being equal, the larger may be the percentage of iron in the tailings.

While this principle is not to be urged in extenuation of careless magnetization and separation, it enables us to view with philosophic calmness an amount of loss in this direction that would cause some of our friends elsewhere to descend into an early grave.

Taking the worst work we have done, three tons of ore to one ton of concentrates, the raw ore carrying 45 per cent. of iron and the concentrates 57 per cent., the cost of the ore would be 75 cents, and adding 50 cents for other expenses we have a ton of 57-per-cent. ore for $1.25. I mention these figures merely by way of illustration, and not as showing the actual cost of the operation.

Crushing the Ore.

The size of the ore most suitable for magnetization is that of a hen's egg. With such pieces the magnetization is thorough even to the center, and when the proper heat has been used there is no sign of louping, or incipient fusion. The ore is of a deep velvety black color. At times the grains of sand are somewhat whitened but for the most part they are coated with a film of the black magnetic oxide. The grains of sand are rounded in the original ore, and in the magnetized ore they are

of the same physical nature. If the heat be too high the sand adheres closely to the ore, incipient fusion having set in, and the subsequent separation is not so good. We have frequently magnetized pieces of ore as large as a cocoanut uniformly to the center; but there is danger in using ore of this size; for when the interior is at a suitable temperature the exterior is apt to be too hot, and there may arise more or less tendency towards louping. We have found that the most suitable heat is a full red, and it is difficult to maintain a large piece of ore throughout at this temperature. The exterior may be just right while the interior of the lump will not be red-hot, and when broken will still be unreduced. It has often happened that a large lump showed a magnetized coating on the outside extending a third or a half of the distance to the center. This coating would be of a dull black color, while the center would still show the original red color of the ore. I have examined a very large number of pieces from the kiln under varying conditions of work, but have not yet seen a piece that was magnetic at the center and non-magnetic on the outside. I mention this because Mr. Barton informed me that when they were magnetizing some small pieces in a coke-oven flue at Pratt mines he found some of them magnetic at the center and non-magnetic on the outside. I have taken particular pains to ascertain if this was the case with the ore magnetized in the Davis-Colby kiln with producer-gas. But although I have examined several hundred pieces of varying sizes and of different degrees of magnetism, I have not observed it. Mr. Barton is far too accurate an observer to make a statement of this kind unless he were sure of it, so I mention it as among the curious phenomena of this process. At present I see no explanation of it, as it has not been possible to demagnetize the ore after it has once been magnetized. That the magnetization should proceed from the center outwards, is certainly not to be expected; and from the fact that in hundreds of pieces, varying in size from a pigeon's egg to a cocoanut, the contrary has been observed, I am inclined to think that the case spoken of by Mr. Barton must have been an isolated one.

It has been our practice to charge the kiln with pieces as nearly uniform in size as possible, so that the gas currents should meet with about the same resistance as they traverse

the space between the combustion-chamber and the draft-flues.

To this end we charged the lumps unmixed with fine ore, the cabs being loaded with forks. A certain amount of fine ore is necessarily made in the kiln itself by the friction of the ore against itself and against the walls of the kiln. This cannot be prevented. The fine ore is as magnetic as the lump, but no more so; and there would not be any serious objection to its presence in the kiln did it not back up into the gas-ports and obstruct the regular flow of gas across and through the lump-ore. It is noteworthy that there has been no louping of the fine ore, this being confined entirely to the lumps.

Charging the Kiln.

This may be done in any convenient manner. At Bessemer, where we had to contend with the disadvantages of circumscribed space, we charged by cab, elevating about 1200 pounds every three minutes on a platform drawn up by wire-rope on a drum. The platform and empty cab are lowered by friction-clutch on the drum when it is thrown out of gear. The kiln holds 110 tons, and is completely filled in a little more than ten hours, starting empty. At the rate at which we ran the kiln there was no difficulty in keeping it full, even when drawing as fast as was possible under the circumstances. In the plant, now being designed, to be built on the mountain in immediate proximity to the ore, we shall charge the ore direct from the mine-cars into the crusher above the kiln, and avoid in this manner all handling of the ore from mine to kiln. An important item of expense will thus be taken away, and there will be no more expense in charging the kiln than in loading a railroad-car, irrespective of the cost of crushing.

The Magnetizing Operation.

The kiln being full of ore, the next step is to heat the ore to such a degree that we may magnetize it. The kiln in use is of the Davis-Colby type, modified to suit the requirements of the case. This kiln is intended for gas-firing, the gas going into the combustion-chamber, taking fire there, and then passing through the ore and into a central vertical chamber occupying the center of the kiln, thence into the stack. The kiln

we are using has a magnetizing-space into which the gas, unmixed with air, may be thrown by a special valve after the ore is sufficiently hot. From this magnetizing-space the unburned gas passes up through the ore, magnetizing it, and then escaping into the stack. We have modified the usual type of this kiln to some extent, but it is not necessary to enter upon this part of the subject at present. Suffice it to say that we found certain changes necessary before we could effect a uniform and thorough magnetization. It may be that we shall have to make other changes before we arrive at a satisfactory position in regard to this matter. It must be understood that we are still experimenting with this kiln, as well as with the process as a whole, so that my reticence on the subject should not be misunderstood. It does not arise from any doubt as to the practicability of the process, but from a hesitation to commit myself as to the form of kiln best adapted for the purpose. The excellent work performed by the Davis-Colby kiln in calcining brown ores, and in desulphurizing certain iron-ores in Pennsylvania, is too well known for me to say a word against it. I regard it as a first-rate kiln for these purposes, and if in its present construction it may not have proved the best for magnetizing, this is to be ascribed rather to lack of experience in this direction than to the kiln itself. In looking into the question of a suitable kiln for magnetizing, we finally decided to build the Davis-Colby, and to modify it as might be necessary after we should have studied the matter more and arrived at a better understanding of what was required.

The kiln is fired with producer-gas, made in the Wellman producer. The coal used is the best quality of Blocton steam-screened, the average composition being given by the following analysis:

	Per cent.
Moisture,	1.00
Volatile and combustible matter,	37.00
Fixed carbon,	59.40
Ash,	2.60
	100.00
Sulphur,	0.90

It is a good gas-coal, delivering about 130,000 cubic feet (8000 pounds) per ton. We used 3 tons per day of twenty-four hours,

so that we had some 390,000 feet of gas for 110 tons of ore, or 3545 feet per ton. This amount of gas would heat the ore to full redness and magnetize it. From four to six hours after starting the fire in the producer, the gas can be ignited all around the kiln at the several igniting-doors. Analyses of the gas from the moment at which it will ignite until it is of a bright orange-red color, and is at its best, are herewith given. The samples were drawn from the producer immediately in front of the pipe conveying the gas into the kiln, care being taken that no air was drawn into the sampler. They were analyzed at once for carbonic acid, oxygen, carbonic oxide, and hydrogen, acetylene not being determined, nor marsh gas, although this latter compound exists in producer-gas to the amount of some 3 per cent. Acetylene seldom occurs in producer-gas beyond a few tenths of one per cent., and may be neglected. As to marsh-gas, it does not seem probable that this gas, in and for itself, can be used to magnetize ore, as the reactions that occur when it is passed over red-hot ore, no other gas being present, are theoretically not such as would lead to a magnetization of the ore. I am unable to speak with confidence on this point, however, as it is a matter of great difficulty to prepare this gas in a state of purity. The ordinary reactions by which it is prepared from sodium acetate yield a gas which is seriously contaminated with hydrogen, rendering it useless for magnetizing-experiments, as the hydrogen is itself a powerful reducing agent, and the magnetic oxide produced by passing marsh-gas from this source over ore would be due to the hydrogen primarily. The question of the effect of pure marsh-gas on red-hot ore is one of scientific rather than of practical moment, as the producer-gas usually employed contains it only to the maximum extent of some 3 per cent.

It can, of course, be prepared pure from zinc methyl; but none of this substance could be procured from dealers in this country, and the question has been dropped for the present.

The effective agents in magnetizing ore are carbonic oxide and hydrogen, and if the producer is operated under the best conditions, there will be enough of these to do the work.

The average content of carbonic oxide while magnetizing was 25 per cent.; of hydrogen, 13 per cent.; of carbonic acid, 6 per cent.; and of oxygen, 0.40 per cent.

Analyses of the Producer-Gas Used.

Carbonic Acid.	Oxygen.	Carbonic Oxide.	Hydrogen.	Remarks.
14.00	None.	8.46	5.93	Bed 4 inches; color dark gray; 1 hour after starting fire.
13.20	None.	6.00	5.60	Bed 6 inches; color dark gray; 2 hours after starting fire.
5.00	0.40	30.80	12.90	Bed 2½ feet; color grayish-red; burns well.
6.80	0.40	25.10	11.90	Bed 3 feet; color orange-yellow; excellent gas.
8.00	None.	24.09	13.84	Bed 4 feet; color orange-red; good gas.
4.30	1.83	22.18	12.63	
6.00	0.60	25.40	14.60	
9.00	0.40	21.40	10.05	Average.

The analyses of the waste gases showed that all the carbonic oxide and the hydrogen were consumed in the kiln.

After the gas has been burning all around for 10 hours the discharging of the kiln can begin. It will be understood that the first 10 or 15 tons, lying at the bottom of the kiln and thus beyond the limit of the heat, are not changed at all and must be sent back as raw ore. This happens only when the kiln is started, for after this portion of ore has been removed so as to give place to ore that has been sufficiently heated and magnetized, all of the ore coming to the shutes has traversed the zone of highly-heated gas and has been exposed to its influence. As the ore is withdrawn from the shutes fresh ore is charged into the kiln, and the operation is continuous. When the ore comes down to the shutes red-hot, the current of gas is changed and instead of passing into the combustion-chamber it is passed into the magnetizing-chamber, from which it passes over the ore unmixed with air and therefore capable of reducing the ferric oxide in the ore into the magnetic oxide. In experimenting with the kiln, we found that even when the gas-valve leading into the combustion-chamber was closed and the valve leading

into the magnetizing-chamber opened, there was still too much air going into the kiln, and we luted the shute-doors with clay. It was extremely difficult to prevent the gas from burning in the ore and thus wasting its reducing power; but by constant attention and keeping the shute-doors well luted, we succeeded in preventing this to a great extent. The reducing-gas was passed over the ore for an hour, when one or two shutes were opened and a cab of ore withdrawn. It was at a full red heat when drawn, and was spread out on the ground to cool. It retained its heat for several hours, but when finally cool enough to handle was of a dull black color, and when coarsely powdered, highly magnetic. The temperature of the kiln, as measured by an Uehling-Steinbart pneumatic pyrometer of the latest construction, varied from 900° to 1350° Fahr., the average being 1100°.

I cannot allow this opportunity to pass without speaking in the highest terms of the most efficient services of Mr. Charles J. Christian in connection with the magnetizing-experiments. He had enjoyed unusual advantages for observing the conduct of the Davis-Colby kiln at Shelby for the past two or three years, and it is to his faithful and unremitting care that a great deal of the success of the experiments is due. In fact, I do not know of any one who could or would have brought to the study of this matter more intelligence or diligence. He suggested many changes in the kiln and in the manner of conducting the operation, which, by trial, were found to be excellent; and this paper could not have been written at this time, except for his constant, cheerful and most effective assistance.

One of the most trying difficulties we experienced was in getting the ore down to the shutes thoroughly and uniformly magnetized. Sometimes the greater part of a cab would be well magnetized while a portion of it taken from the same shute at the same time, would not be magnetic at all. This was found to be due to the fact that it had not been exposed to the gas for a sufficient length of time. As proof of this, I took some of the gas that was going into the kiln and some of the non-magnetized ore from a cab, heated the ore to a full redness in a glass tube and passed the gas over it. It became magnetic in a few moments. After this, we allowed the gas to pass over the ore for a longer time, and found that when the shutes were well

luted and the air excluded, we obtained better results. One thing was proved to our entire satisfaction, viz., that when the ore was exposed for a sufficient length of time at a full red heat to a current of producer-gas it became highly magnetic, and that this effect was to a considerable extent independent of the size of the lumps. The difficulty already alluded to, the tendency of the larger lumps to loup, was hard to overcome. The outside of these pieces would be magnetic while the interior would not be changed at all, or at best would exhibit very feeble magnetism.

Now and then, a lump as large as a cocoa-nut would come down in a very satisfactory condition, but on the whole it was found desirable to exclude these large lumps from the kiln and to use ore that was of the size of a hen's egg. Another serious difficulty was in the irregular manner in which the ore came down to the shutes. In a kiln of this construction it is very difficult to get a uniform heat all round. At times the kiln would be hot enough on one side, too hot on another and too cool somewhere else. When it became too hot on one side, there was nothing to do but to draw ore from the shutes on that side and let the ore descend until the normal heat was restored. This naturally disturbed the course of the operation elsewhere in the kilns, and had a tendency towards allowing insufficiently-magnetized ore to come down to the shutes and in a measure to occupy a space outside of the area of magnetization. When the operation was proceeding satisfactorily, we got from the kiln 110 tons of ore per day of 24 hours, and worked in this way for several weeks. A part of the ore was magnetic, and a part was not. It was culled for separation. The separating machine could not treat half of the ore that was magnetized every day; and the remainder was sent direct to the furnace without separation.

Concentration of the Magnetized Ore.

This was effected over a Hoffman separator, which we were enabled to use by permission of the Magnetic Separator Company, of Troy, N. Y. It was actuated by a 3 K. W. dynamo, Thomson-Houston system, and was run under 10 to 15 amperes and 110 volts, the speed of the belt being 66 revolutions per minute. The magnetized ore was first crushed in a No. 3

Gates crusher, screened over a revolving screen of 8 meshes per linear inch, the heads from the screen going into a pair of rolls and thence into the conveyor with the tails from the screen, and so on to the bin above the separator. Between the end of the conveyor and the bin there was another screen of quarter-mesh size to remove the small lumps that jumped the rolls or passed down between the ends of the rolls and the housing. All the material going to the separator passed this screen and nearly all passed a screen of 8 meshes per linear inch.

The fineness of this material is given in the following table:

Fineness of Material Going to the Separator.

	Per cent.
Left on 8-mesh screen,	3.00
Through 8- " " and on 10-mesh,	6.50
" 10- and on 20-mesh,	28.50
" 20- " 30- "	31.50
" 30- " 40- "	6.50
" 40- " 50- "	9.50
" 50- " 60- "	2.50
" 60- " 70- "	3.50
" 70- " 80- "	None.
" 70- " 100- "	3.50
" 100-mesh,	5.00

This represents the average fineness of the material sent to the separator during the course of the experiments, as several determinations were made from time to time.

It was not found practicable to run the separator at a greater speed than would give about 700 pounds of heads per hour, as we had difficulty in disposing of our tailings in greater quantity than this, owing to the confined space in which we had to work. The separation was attended by a good deal of dust until we regulated the feed to this point, and even then it was far from pleasant. Special care has been taken of this in the plans for the alteration of the plant; and we shall remove the dust by means of an air-blast. The average content of iron in the ore sent to the separator was 45 per cent. and of silica 30 per cent. The average content of iron in the heads was 58.86 per cent. and of silica 11.51 per cent.; in the middlings 51.12 per cent. of iron and 21 per cent. of silica.

At the very start we found that some portions of the ore were more highly magnetic than others and that the less magnetic material manifested a strong tendency to go into the tails and

not into the middlings. In other words, the tails contained magnetic ore that should have gone either into the heads or at any rate into the middlings. Adjustment of the machine and changes of the amperage enabled us to correct this to some extent; but we did not succeed in doing away with it entirely, and throughout the entire course of the work we were troubled with incomplete separation. Repassing the tails over the machine always resulted in obtaining more heads and middlings than in the first pass, and we finally concluded that it was practically impossible to get even tolerable tails by one pass. To this conclusion it seems that all have come who have tried magnetic separation, even of highly magnetic natural magnetite, viz., that it is in all cases advisable to use two machines or, better still, two drums, and to pass the middlings and tails from the first to the second, increasing it may be the amperage on the second machine or drum, and, perhaps, also regrinding the material from the first machine before sending it to the second. As by far the greater part of the expense in the magnetic separation of ore is incurred before the ore is sent to the separator, the additional expense of sending it to another machine, even should it be reground, is comparatively slight. It may be of interest to some to know the distribution of the iron and the silica in the heads according to the fineness. I give, therefore, in the following table some analyses covering this point. Numerous analyses have been made to show just where the best ore was, and if finer grinding would enable us to improve the quality of the heads. From these I select the following:

Analyses of Heads According to Fineness.

Original Ore: Insoluble, 28 per cent.; Iron, 44 per cent.

	Per cent.	Insoluble.	Iron.
Left on 8-mesh screen, ...	3.00	12.76	63.20
Through 8- on 10-mesh screen, .	6.50	12.50	62.70
" 10- " 20- "	28.50	13.00	61.30
" 20- " 30- "	31.50	13.40	60.00
" 30- " 40- "	6.50	13.70	60.30
" 40- " 50- "	9.50	15.40	58.25
" 50- " 60- "	2.50	13.90	60.80
" 60- " 70- "	3.50	14.00	60.70
" 70- " 80- "	None.	......	......
" 70- " 100- "	3.50	14.70	60.00
" 100-mesh, ...	5.00	16.10	57.00
Average, ...		13.94	60.42

It might be inferred from these analyses that the amount of iron decreased with the fineness; but that this is not always the case will be apparent from the following analyses representing the heads at another period of the work:

Analyses of Heads According to Fineness.

Original Ore: Insoluble, 32 per cent.; Iron, 40 per cent.

	Per cent.	Insoluble.	Iron.
Left on 10-mesh screen,	2.90	12.65	59.15
Through 10- on 20-mesh,	18.30	12.58	59.09
" 20- " 30- "	21.70	12.72	59.25
" 30- " 40- "	10.00	12.65	59.20
" 40- " 50- "	10.00	12.40	59.48
" 50- " 60- "	13.30	11.05	61.73
" 60- " 70- "	8.50	11.08	61.80
" 70- " 80- "	None.	......	......
" 70- " 100- "	10.00	11.45	61.40
" 100-mesh,	5.30	10.80	62.00
Average,		11.93	60.33

There does not seem to be any fixed rule as to this matter; sometimes the percentage of iron increases with the fineness and sometimes it does not. It may be chargeable to the nature of the ore, if easily pulverized or not, the degree of magnetism in the ore (about which very little is known, whether the ore be natural or artificial magnetite); the intensity of the current; the speed of the machine; or a combination of these causes. It is still under investigation, and we hope to be able to throw some light on it later.

I have spoken, in passing, of the degree of magnetism imparted to the ore. This is a subject upon which very little is positively known. In some recent articles in the *Engineering and Mining Journal*, I have endeavored to point out some facts in connection with this most interesting and, to us, important matter. It was remarked there that perhaps it made not so much difference to those who were engaged in the separation of natural magnetite, where there was no effort to increase the magnetism of the ore. As practically all of the separation undertaken in this country has been based on natural magnetites, the question of the degree of magnetism possessed by such ores has not received very much attention. It is true that it has an important bearing even on the separation of natural mag-

netites, for the fact that the degree of magnetism does vary is proved by the production of heads and middlings over the same machine and under the same conditions. But when we come to the magnetization of non-magnetic ores, the degree of magnetism that may be imparted to them becomes of the most vital importance. If it is possible to effect a commercial separation of such artificial magnetite when the amount of iron as ferrous oxide is only 10 per cent., why increase it to 20 per cent.? The consumption of gas per ton of ore is much greater when it becomes necessary to produce the higher article than it is for the lower, and as every cubic foot of gas, after the ore has been raised to the proper temperature, is supposed to convert a certain amount of ferric into ferrous oxide, we should be wasting gas in making the 20-per-cent. stuff if the 10 per cent. would answer the purpose as well.

For instance, we found that some of the material we were concentrating contained only 8.50 per cent. of iron as ferrous oxide, while another portion contained 25.80 per cent., and yet we could perceive no striking difference in the degree of magnetism as measured by the concentration over the machine. The heads were as good when using the material containing the less amount of ferrous oxide as when using that containing more ferrous oxide. In this connection one recalls the statement made by Mr. T. R. Woodbridge,* that he had seen pieces of Michigan hematite, containing less than 0.20 per cent. of ferrous oxide, magnetic, one fragment weighing about one-third of a gramme jumping one-eighth of an inch to a magnet run by two one-half gallon Bunsen cells. The question of the degree of magnetism in artificial magnetite is of a great deal of interest to us, and I hope by the next meeting of the Institute we shall be able to communicate something of value in the elucidation of it. Such questions require long consideration before they can be satisfactorily answered, and although we have some data bearing on the subject, it is, perhaps, better to withhold them until they can be added to and brought into practical shape.

Another matter which possesses for us unusual interest is the possibility of the magnetized ore becoming demagnetized, either

* *Trans.*, xx., 578.

in the kiln through the irregular working or afterwards, through the action of the air or from any other cause that might come into play. We have conducted a number of experiments on this point. It is stated in most of the books on mineralogy that natural magnetite can be demagnetized by heating in an oxidizing flame. This may be true of the samples examined, and doubtless was the case in the instance quoted. But I have heated fragments of highly magnetic natural magnetite in an intensely oxidizing flame for thirty minutes without affecting the magnetism, so far as could be observed. At a dull red heat in an oxidizing flame some fragments lost their magnetism but recovered it at very high temperatures. This tendency towards magnetism at high temperatures was observed in the case of sesquioxide of iron by H. Rose,* who states that on heating this compound in a porcelain kiln it lost oxygen and became magnetic. In like manner magnetite may absorb oxygen and lose its magnetic qualities; regaining them, however, at very high temperatures by loss of oxygen and reconversion into ferrous-ferric oxide.

If artificial magnetite should become demagnetized before separation, we should lose the cost of the magnetizing, and be in pretty much the same condition as when the operation was begun. If it should become demagnetized after separation, we should lose percentage of iron in the proportion in which oxygen was absorbed, as the heads would weigh more, but carry less iron per unit of weight, than when they were made.

To test this matter, several grammes of heads in which the total iron and that present as the two oxides, ferrous and ferric, had been determined, were exposed to a current of pure, dry oxygen in a glass tube for different periods of time, ranging from ten minutes to one hour, at a full red heat. On cooling in the tube in the current of gas, the ore was found to have lost its black color, and to have become reddish, but it was still magnetic, and did not seem to differ in this respect from the original. This effect was observed after treatment for thirty minutes, and the amount of iron as ferrous oxide fell from 19 per cent. to 12 per cent. On continuing the treatment for thirty minutes longer there was no further change in color, but

* Quoted by Percy, *Iron and Steel*, p. 16.

the amount of iron as ferrous oxide fell to 8.50 per cent. without seemingly affecting the magnetism.

Treatment for two hours with the oxygen gas failed to convert all of the ferrous into ferric oxide, or to deprive the ore of its magnetic qualities. The ore was not demagnetized by a current of superheated vapor of water or carbonic acid, or by a mixture of these. The conclusion that the magnetism imparted to the ore is of a permanent character, seems to be well founded. At any rate, none of the gases present in the kiln, during or after magnetizing, seem to have the power of demagnetizing ore once magnetized, and we may regard it as practically impossible to interfere seriously with the magnetism acquired in this process.

This process differs radically from that used in Savoy, reference to which has already been made. In that, the purpose is to deprive the ferrous carbonate of its carbonic acid in an atmosphere of reducing action, in order that all of the ferrous oxide left on the removal of the carbonic acid may not be converted into ferric oxide. Should this be the case, the resulting material would not be magnetic, and, of course, the separation by means of the current would not be applicable. On calcining ferrous carbonate in an atmosphere in which there was no oxygen, we should be able to prepare ferrous oxide. If oxygen is present, we obtain a mixture of ferrous and ferric oxides, which is magnetic; and if there is excess of oxygen, or just enough to oxidize the ferrous oxide, we lose the magnetic quality we seek. In other words, the Savoy process is one of partial oxidation, this process is one of partial reduction. The chemical principles involved are entirely different; for if the Savoy process be continued to its legitimate conclusion, viz., complete oxidation, we should not have any magnetic ore left. If, on the other hand, this process be continued to its natural conclusion, viz., complete reduction, we should obtain a material even more magnetic than the ore—that is, metallic iron. But it does not follow that the reagents employed should be different. The same gas used in the Savoy process can also be used in this. Producer-gas is used there and producer-gas is used here, but for different purposes. There it is used to prevent oxidation beyond the point at which the ore ceases to be magnetic; here it is used to effect a partial reduction, and

the exclusion of air, or oxygen, is as important in the one case as in the other. The presence of silica does not interfere seriously with the carrying out of either process, for the chemical reactions of the two are independent of this substance. It is true that when the heat is so high as to induce incipient fusion, the separation of the more ferruginous portion of the ore from the more sandy portions is not so complete; but this is an accidental, not an essential, part of the process, and in discussing it from the standpoint of the metallurgical chemist, we may consider the silica as not in any wise affecting the magnetization. It may, and does, affect the separation; but this is a point not now claiming our attention. In so far as it interposes an impervious cementing-material to the action of the reducing gas, preventing the action of the gas on a particle of ore, silica may play a part of some importance in the process; but the cement is not siliceous; it is ferruginous. This is proved not only by the fact that on digesting the raw ore in dilute hydrochloric acid for some days it yields almost pure sand, but also by the further fact that on calcining the lime-ore and slaking it in the air or in water, it falls to coarse powder, and the grains of sand can be picked out by hand or separated over a magnetic machine. It is one of the good qualities of the ore we propose to concentrate that the sand grains are all more or less rounded. The wear on shutes, conveyors, etc., is, therefore, of less moment than if they were angular or had sharp edges.

So far, nothing has been said as to the removal of phosphorus. This element is present in the ore to about 0.30 per cent., but it is not removed in the separation. It seems to be present as phosphate of lime, entirely amorphous, and most intimately mixed with the iron. We have not been able to remove it, or even to diminish it to any considerable extent. No matter how finely the ore is ground, the heads still carry more phosphorus than is allowed in Bessemer ore. It can be entirely removed by chemical means, and brought from 0.30 to 0.008 per cent. at one operation. It has been found that dilute sulphuric acid will dissolve out the phosphorus from the heads without affecting the content of iron seriously, and in this manner heads carrying from 58 per cent. to 60 per cent. of iron and 0.008 per cent. of phosphorus have been prepared.

A word now as to the cost of carrying out this process on a

scale, let us say, of 100 tons of raw ore per day of twenty-four hours. We will assume that the plant is erected on the mountain in immediate proximity to the ore, and that the gravity system is employed for conveying the ore from the mine to the kiln and from the kiln through the various operations until the concentrates are loaded on the cars. We will allow, also, that it requires 3 tons of raw ore to 1 ton of concentrates carrying 55 per cent. of iron, and that the yield of such concentrates from one kiln is 27 tons per day of 24 hours. In other words, we allow that from a kiln holding 100 tons of raw ore we obtain daily 81 tons of magnetized ore fit for separation. The cost of producing 1 ton of concentrates of 55 per cent. iron will be about as follows:

3 tons of raw ore, at 25 cents,	$0.75
Crushing, including labor,	0.05
Discharging kiln,	0.06
Crushing, rolling and screening,	0.05
Separating, and disposing of tailings,	0.05
Superintendence,	0.04
Night foreman,	0.02
Engineers,	0.04
3 tons of coal for producer, at $1.25,	0.04
3 tons of coal for boilers,	0.04
Oil, supplies, etc.,	0.01
	$1.15

These are the estimates that have been made from our experience with the process at Bessemer, where we had to work under unfavorable conditions, and where the cost per ton of 55-per-cent. concentrates was 40 cents higher than the above figures. If we are able to increase the percentage of iron in the concentrates, as we expect to do, the cost per ton will be lessened accordingly. On the other hand, should we not be able to do this, but have to allow for 3 tons of raw ore per ton of 55-per-cent. concentrates, as above, the cost will not vary much from that given, viz., $1.15.

We come now to the question, Is a ton of 55-per-cent. ore of the fineness already given, worth $1.15 at the works, or $1.30 at the furnace? In valuing an ore for furnace-practice, two methods may be used, the one based on the nature of the iron desired to be made from it, whether special high-grade Bessemer or basic open-hearth; the other, disregarding this

feature of the question, is based on the ordinary grades of foundry-, forge- and mill-irons made in the district. Both methods are in common use, and both are independent of the reducibility of the ore, this factor of the question not being generally considered.

The matter, then, narrows down to the question as to whether this ore, under the conditions now maintaining in the Birmingham district, is worth to the furnaces $1.30 per ton, delivered.

This may, perhaps, be answered to the best advantage if we inquire as to its value if it alone were to be used in the furnace. As a matter of fact, unless it be made into briquettes, eggettes, or other suitable shape, by means of some binding material, it can not be thus used; but for the purpose of this calculation we may assume that it can.

We will assume that the limestone to be employed as flux contains 3 per cent. of silica, that the coke used as fuel contains 10 per cent. of ash, or 5 per cent of silica, and that the ore contains 55 per cent. of iron and 13 per cent. of silica. What will it cost to make a ton of iron with these ingredients, allowing 2400 pounds of coke per ton of iron?

1.82 tons of ore at $1.30,	$2.36
1.20 tons of coke at $1.75,	2.10
0.66 ton of stone at 0.60,	0.39
	$4.85

This cost is, of course, to be taken as representing the cost of the materials entering into a ton of iron, and does not include labor costs, repairs and interest, and is based on ordinary foundry-irons with slag carrying 35 per cent. of silica. With this result no one who is familiar with the practice in the district can doubt that the furnaces can afford to pay $1.30 for these concentrates. But it will be said that at this price there would be no profit to those engaged in making the concentrates. If the cost at the works is $1.15 and freight 15 cents, where is the profit in selling at $1.30? It would not exist, as a matter of course; and if the business can not do better than this, it can not maintain itself at all.

Either the furnaces must pay more for the ore or it must be made at a less cost, if one is to go into the open market and

sell the concentrates profitably. Under the present circumstances, it is not proposed to sell the concentrates in the open market but to use them in the furnaces belonging to the company, and the value of them is established by the foregoing calculation as well as by experience elsewhere with somewhat similar material.

It seems to me that the concentrates will occupy about the position of brown ore with respect to the amount used in the furnace. It may be that it will not be practicable to burden a furnace exclusively with such fine stuff, and in this particular the comparison with brown ore breaks down; but to strengthen the iron, lessen the cinder-flow and increase the make, the two stand pretty much on the same plane. It has happened in the district that brown ore of excellent quality was sold for $1.50 delivered, although the usual price is considerably below this figure, and the same price would be paid for concentrates of the same or better grade.

At first blush it would seem that to propose to a furnace to pay even $1.50 for ore when it has been paying from 50 to 60 cents, would be to invite ridicule; but there is no fact more firmly established than this, viz., the value of an ore for blast-furnace purposes increases very rapidly with the increase of iron and the decrease of silica. Less fuel and less flux are required, and consequently less labor in handling the material going into a ton of iron. The stock will be more uniform, and the *bête noire* of Southern furnaces, stock of irregular composition, will be rendered less terrible.

Aside, however, from considerations affecting the cost of making iron, with or without these concentrates, in the Birmingham district, the success of the process will bring into use very large deposits of soft ore now practically worthless, and enable the owners of such ore-lands to realize more on their investment than they could otherwise hope to do. The supply of the better grades of soft ore is not indefinitely great, and even where the quality of the seams justifies mining, with the exception of some narrow seams of high-grade ore, very little more than half the seam is now being taken. It follows that the original cost of the ore-lands must be doubled if the lower part of the ore is not used, and in charging off the cost of the land this fact must be considered. If this process will enable

us to utilize the whole seam, top, middle and bottom, all along the Red Mountain, the supply of soft ore is very greatly increased and the cost of making iron will continue lower than if we had to mine ore under ground.

I am not yet prepared to assert positively that the process is all that could be desired. There still remain some important questions to be solved before such an assertion can be made. But after studying the matter in all its bearings for more than two years, and after experimenting with it on a working-scale for several months, I feel warranted in saying that it is full of promise for the district and might with advantage be applied elsewhere, especially where the first cost of the ore is low.

This paper has been prepared, not for the purpose of "booming" the process, but merely to set before those who may be interested in such matters the first steps in what we have reason to believe will prove of some importance to iron-masters in general, and to the Southern iron-trade in particular.

The Florida Pebble-Phosphates.

BY E. W. CODINGTON, BARTOW, FLORIDA.

(Florida Meeting, March, 1895.)

Descriptive.

The pebble-phosphates of Florida occur in a district roughly bounded on the north by the 28th parallel, on the east by an irregular line running a few miles east of Peace river and on the west by the Gulf of Mexico, and covering an area of about 4000 square miles. There seems to be no reason why the deposits should not extend farther south along the gulf coast. That section remains unprospected, transportation-facilities being non-existent.

It is not intended in this paper to discuss the geology of these formations; but a word as to their probable origin will be perhaps the best method of giving a correct idea of their present position. Starting with a low, flat surface, with even less of superficial undulation than appears at present, an im-

mense wash (probably the Gulf Stream, which closely hugs the present coast-line) has carried the breakage of the hard-rock region lying to the northward (contaminated slightly by the wash from middle Georgia) and deposited its burden along the reach of coast-line. That the water-action was continuous, rapid and turbulent, is evidenced by the comparative absence of stratum-lines and the varying nature of the deposit. Winds, waves, streams, currents and eddies have all played their parts. In places, the pebble is nearly, even wholly absent and in other places there is scarcely more of the matrix than would fill the interstices; with all grades of richness between these two extremes. The matrix is sometimes (but rarely) little more than clear sand. In such cases there is evidence of later surface-wash or filtration, notably in the absence of soft pebble. Generally, the matrix contains a more or less intractable mixture of clay sediment, in which Dr. Pratt, of Atlanta, in analyzing a mass-sample, discovered an increment of fish-slime.

The presence of fish-slime may or may not be a general characteristic. So far as now known, it is a matter of no consequence in practical working, although it might be of some importance in scientific research.

The wash from middle Georgia is indicated by an admixture of quartz pebble found throughout the whole district, sometimes in inappreciable quantities and sometimes forming a serious contamination of the phosphate-rock. Vivianite occurs in a few instances as an appreciable contamination, but is not general.

The pebble-phosphate deposit, as thus crudely described, is overlain by a formation mainly of drift-sand, varying in depth from nothing into the region of conjecture. The drill has penetrated this deposit at depths of more than 100 feet; and, almost universally, the depth of overburden is closely related to the present surface-conformation, small areas only being considered.

Considerable areas are found at intervals, overlain by sand-conglomerate, generally broken into boulders, but occasionally occurring as a distinct stratification, with wave-action plainly marked. This occurrence of conglomerate is a matter of interest as affecting methods and cost of exploitation, and will be again referred to.

The river-pebble, as a distinctive formation, occurs only by reason of a running stream over-running and cutting through a land-deposit. It has thus been displaced from its original matrix; the soft pebble has been destroyed by abrasion; and it now occurs in the beds and bars of streams as a simple mixture of sand and gravel.

There seem to be no surface-indications of the existence of beds of pebble-phosphate. If, as claimed by some, the flora has been improved by underlying phosphate near the surface, it is only in that slight degree which becomes discernible after one thinks it ought to be so. Some prospectors maintain that large areas of saw-grass promise sure results, but this is not a fact except in the matter of accessibility, saw-grass land being always low-lying, and therefore indicating shallow cover, if there be a deposit under it at all.

Overburden.

For the purposes of exploitation, overburden may be divided into two kinds, viz., that which contains sand-conglomerate and that which does not. The first is objectionable on account of the necessity for its thorough removal before the phosphate stratum can be mined. The dividing line is so plainly marked that but little of the good rock need be lost; but the necessity for avoiding contamination makes this one of the most critical points of mining operations wherever the conglomerate occurs. It seldom fails, also, to present some interesting and exasperating problems as to processes for removal. Given a depth of three feet or less, the natural impulse of the operator is towards the wheelbarrow or scraper. In opening a mine the lightest overburden is naturally made the starting-point; and reluctance or inability to stand the cost of additional plant often keeps the wheelbarrow employed long after its day of usefulness has passed. Allowing 20 cents per yard for removing overburden with scrapers or wheelbarrows (the average cost has been rather more than less), and 5 cents per yard for the steam-shovel, the cost of removing 12 feet by the faster method only equals that of 3 feet by the slower, and in either case has added 77 cents per ton to the cost of mining a deposit running 5000 tons to the acre.

It is our purpose to describe and not to criticize, but it is not

easy to pass without remark the present unskilful handling of the conglomerate overburden. The cable-tram, which should play an important part at most of these mines, is entirely absent, and the overburden is piled in great masses over rich deposits, which the miner cannot work until this mass of *débris* is moved again.

That oldest of all engineering problems, "how to move dirt at the least cost," is strikingly *en evidence* here. Apparently simple, any miner assumes to solve it off-hand, and yet it is the rock on which more good engineers go to pieces than on any other. It cannot be taught in the schools, for it never appears twice in the same form. The only rule is the rule of thumb; but one principle may be applied with unerring certainty, viz., with negro labor at $1.25 per day, there is no room for the negro on ground that might be occupied by a machine.

Where no sand-conglomerate occurs, the disposal of over-burden is of little importance except that it adds to the volume of the *débris*. If the matrix is tough and intractable, the sand over-burden will (within certain limits, say 5 or 6 feet in depth) assist in washing, and may be considered a benefit. In hydraulic mining, it could, if desired, be easily screened overboard without passing through the washer at all, so that, at the worst, it involves no more additional expense than a little larger centrifugal pump and the power to drive it.

Phosphate-Conglomerate.

In deposits that are elevated above the present water-line, there frequently occurs a formation (probably caused by the infiltration of chalybeate water) of what may be termed phosphate-conglomerate. The iron in solution, having been deposited, has furnished the necessary bond; and the result is an indurated deposit composed of all the ingredients of the ordinary pebble-deposit, but with iron oxide in excess. This class of rock always holds a certain advantage in being easily mined (not milled), lying as it does above water-level. Up to the present, however, it has never been profitably handled. Some efforts have been made (notably at the Moore & Tatum mine, 3 miles south of Bartow) to prepare this rock for market by a dry process. The Moore & Tatum deposit is a remarkably fine example of this formation; and it is difficult to see why mining

operations thereon should not be profitable, except that they, so far, lack the vital element, success. The methods employed were, first, disintegration by coarse crushing. The stuff being sufficiently dry to prevent adhesion, it came from the crusher in the form of fine gravel; it was then passed over hot plates by means of a drag-conveyor and screened into the storage-bins. The material thus produced reached easy marketable grade in bone-phosphate of lime, but carried from 5 to 9 per cent. of iron and alumina. It seems certain that a self-scouring process, something in the nature of the revolving barrel, would reduce materially the iron and alumina, and ought to produce, at a low cost, rock that would fill the requirements of the domestic market. Doubtless the use of a rotary dryer, in place of the furnace and conveyor, would go a long way towards producing the desired effect.

Another method of working this class of rock has been in use for several years at the Whitaker Phosphate Company's works at Homeland. The rock is dried and then ground by self-attrition over an air-blast which deposits the different grades (in point of fineness) at different points in the discharge, the sand being left as the residuum. The sedimentary matter which contains the iron and alumina is not eliminated. This product has been sold as a natural fertilizer, but its status as such remains unfixed. It seems to produce the same results as the soft phosphates used in a similar manner, and it is fairly well settled that it has real value as a fertilizer for leguminous plants. The usual practice of acidulation cannot be introduced here on account of the excess of iron and alumina.

Mining.

There are two distinctive methods of mining land-pebble, which for convenience may be termed "dredging" and "hydraulic."

The first naturally applies to such mines as raise the rock from under water, and the last to such as work in dry pits. It is scarcely necessary to add that dredging is sometimes a matter of necessity and not of choice. Some car-mining has also been done in pits that could be drained, but such work has been found unprofitable, and has been wholly abandoned.

In dredge-mining, all types of machines have been used.

We need not here discuss the relative merits of the "dipper," the "clam-shell" and the "elevator," as their adaptability is largely governed by location and environment. Whichever one is used, the function is simply that of raising the stuff from under water and passing it to the wash-boat alongside the dredge.

All washing is done by some attrition-process, the log-washer in some form being in almost universal use. The amount of manipulation required for clean washing varies considerably in different deposits, or even in different parts of the same deposit. The *débris* is carried directly overboard, unless the necessity for preserving navigable water forces a partial departure from this practice.

The washed material is taken to the mill for drying and storing. This is a process common to all methods of mining, and will be described later.

Hydraulic mining presents some points of advantage which will probably always hold it in favor wherever it is practicable, that is, wherever the pit can be pumped dry with a reasonable amount of pump-capacity. The term fairly describes the process, which is simply to play giant-nozzle streams, supplied by strong hydraulic pumps, against the wall of the pit. This rapidly disintegrates the mass and carries the loose material to the sump-hole, where it is taken up by a centrifugal pump and carried to the washer.

The advantages are: (1) the washing is largely done in the pit, and only a small washer is needed on the barge or at the mill; also, the amount of washing in the pit can be regulated to meet the work of the washer, thus making it easy to handle material of varying tractability always to the best advantage; (2) the work being always in sight, the deposit can be worked clean without taking up worthless material; (3) it is probably the cheapest method of raising the rock; (4) it is less destructive of soft pebble.

It may be remarked *en passant* that soft pebble indurates rapidly by exposure and heat, so that whatever gets through the washer is not lost in any subsequent screening.

The best processes now in use for mining and preparing the rock leave little room for future economies. The plants now in operation are free from many former errors of construction and

adjustment of machinery; and the crude, tentative methods, which characterized the early days of mining and sunk out of sight an enormous amount of good capital, have disappeared. There is still one line, however, along which improvement is to be desired, and that is the salvage of the soft pebble which is always present in the land-deposits (it has been destroyed by abrasion in the rivers), and which, in some otherwise good mines, forms so large a percentage as to make the working unprofitable. This soft pebble is destroyed in washing, and passes off with the *débris*, which must of necessity be the case in any attrition-process of washing. As a rough rule, it may be said that the more intractable the matrix, the more soft rock is contained therein; so that deposits which might be of great value if the full amount of phosphate-pebble could be reclaimed, become worthless because the value is destroyed in washing.

Some interesting experiments by Mr. George Guild, with the direct application of steam as a solvent, have resulted in the patenting of a device known as the Guild steam-washer. The machine consists of a cylinder in which is a slowly revolving finger-shaft. The mass of material is dumped into the cylinder, which is then closed and steam is admitted. The steam is promptly diffused by means of the finger-shaft, and in a few minutes the whole mass is heated and assumes the consistency of thin mortar. It is then ejected and passed rapidly over a screen, under jets of water, when it is found that the phosphate pebble remains intact while the matrix is thoroughly dissolved and drops readily through the screen.

Many samples have been tested in a small machine with the most satisfactory results. In one case the yield was increased from 8 per cent. to 25 per cent. of pebble to the mass. No working-plant has yet been built, but the invention promises a bright future for deposits which must remain worthless under present methods of manipulation. The soft pebble being generally of the highest grade, it is quite possible that the best mines of the future may be in deposits now condemned as inferior or valueless.

In mining pebble-phosphate, it is very clear that water plays an important part. It should be abundant and (at least a portion of it) should also be clean, for, in the final rinsing, the use of muddy water would lower the grade and add iron and

alumina in damaging quantities. While surface-streams are sometimes lacking in close proximity to good deposits, subterranean streams seem to be nearly universal.

From Brunswick, Georgia, south along the east coast, and across the peninsula below the 28th parallel until overlying coral is reached at the southern end of the peninsula, I know of no instance of failure to get a water-supply within 400 feet of the surface, by drilling. Throughout the pebble-region, water is almost always found within 200 feet and sometimes within 50 to 60 feet. The upper streams are almost always nearly pure, while the lower veins (from 300 feet and upwards in depth) are more or less impregnated with sulphuretted hydrogen.

Drying.

Various appliances for drying pebble-rock have been used experimentally and discarded. Among these are, notably, various forms of brick chimneys, which proved too destructible; and conveying through heated troughs, which proved too inefficient and expensive.

The rotary dryer is now in almost universal use. The best are built of wrought-iron, about 25 feet long by 36 inches diameter, revolving on two or three sets of guide-wheels placed at third- or quarter-length distances; the cylinder being driven from the middle by a gear-flange and pinion, or a sprocket-flange and chain. The cylinder is rifle-flanged inside, which serves the double purpose of carrying forward the rock and keeping it constantly stirred. The furnace is at the discharge-end and the feed under the smoke-stack. The fire passes directly through the cylinder, thus furnishing the necessary draft to carry off the steam.

The rock must be dried to below 2 per cent. of moisture.

Present Condition of the Industry.

A rough estimate would give an area of 5000 acres with an average yield of 5000 tons per acre, closely available at present market-value. It is evident that any increase of price would bring a proportionate amount of territory into profitable working. A decrease in carrying-charges would work the same result; the present carriage and terminal charges are exorbitant. The cost of removing 12 feet of conglomerate overburden from

an acre of land, could be taken off from the cost of transporting 5000 tons from the mine to the ship, and still leave the carrier more than a fair profit. This state of affairs cannot always hold, for the miners are doubtless paying in excess of cost of service, enough to pay for their own railroad and terminal inside of two years, at the present rate of working.

Under existing conditions, and always assuming the use of the best mechanical appliances (which is by no means the case with all plants now in operation) a workable deposit must fill approximately the following specifications, viz., on the basis of a phosphate-stratum of 12 feet in depth, averaging 600 pounds of dried pebble to the cubic yard of mass, the overburden should not exceed 3 feet if it contain sand-conglomerate, or 12 feet if free from the conglomerate. The rock should not run below 65 per cent. phosphate of lime in cargo-samples, nor above 3 per cent. combined oxide of iron and alumina; and the location should be such as to present no special mining difficulties.

The amount that could be mined so as to be manufactured and used on land profitably, at the present price of farm-products, is another matter entirely, and any attempt to put it into figures might easily leave ground for criticism as to the number, rather than the size of the digits used. It is enough to say that lack of phosphoric acid will not render our country incapable of supporting its population until long after we have ceased to take any personal interest in the matter.

Biographical Notice of Moritz Ferdinand Gaetzschmann.

BY R. W. RAYMOND, NEW YORK CITY.

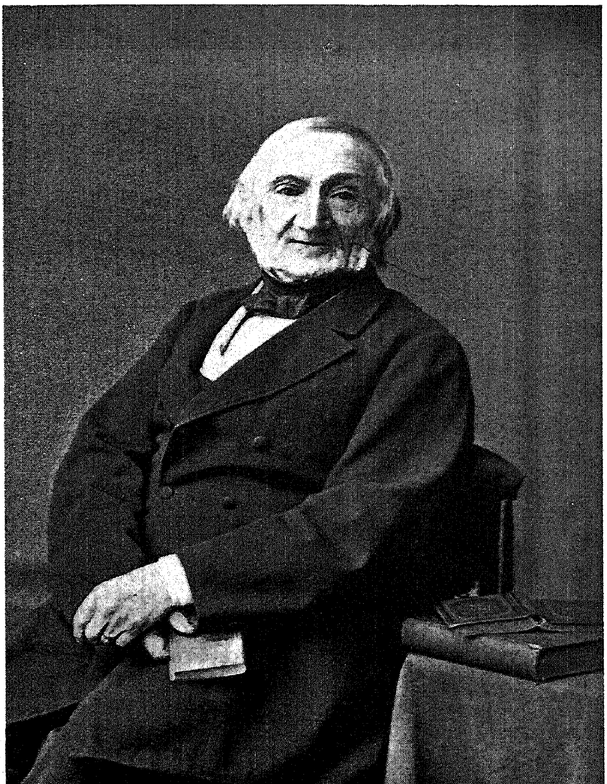
(Florida Meeting, March, 1895.)

By the death of Bergrath Prof. M. F. Gaetzschmann, which occurred on the 18th of February, in the 95th year of his age, at his lifelong home in Freiberg, Saxony, the Institute loses one of its earliest and most eminent honorary members, and the profession of mining and engineering one of its most influential and useful representatives.

Born with the century, at Leipzig, he entered the Freiberg Mining Academy as a student in 1821. After graduation he became a lecturer and instructor of mine-surveying, and was afterwards a mining official at Schneeberg. In 1834 he was called again to Freiberg, where he was, until 1836, lecturer on mine-engineering, and from 1836 to 1871 occupied the chair as Professor in that department, retiring in the latter year from active labor, though not surrendering his interest in the work to which his life had been devoted, or in the numerous pupils whom he had trained. In 1862 he received, in recognition of his distinguished services, the title of Bergrath.

Besides his continuous labors as a teacher (of which I shall have more to say), he enriched the technical literature of mining by many valuable contributions, most of which were intended to be parts of a complete text-book, never finished. The first was an essay on mine-masonry, which appeared in 1831. In 1846 he published what was to be the third part of his great work, a treatise on methods of exploitation (*Bergmännische Gewinnungsarbeiten*); and in 1856 the first part, namely, a volume devoted to the subject of prospecting (*Die Auf-und Untersuchung von Lagerstätten nutzbarer Mineralien*). This book has remained for nearly 40 years unsurpassed as a trustworthy and helpful authority. Another part of the projected cyclopædia was completed in 1872, and dealt with the mechanical concentration of ores. Of his various other literary activities, as editor of the Freiberg *Jahrbuch*, as a director of the Academy library, and as author of sundry technical and historical monographs, I will not attempt to give a full account. But his statistical compilation in two volumes (1852) of the product of the Freiberg district from 1530 to 1850, and his admirable little pocket glossary (1852) of mining terms, deserve special notice. And I must not omit to mention the fact that he was the first to conduct the admirable practical preparatory course in mining, established at Freiberg in 1846. This position, in addition to his work as a lecturer and instructor, brought him into personal relations with the students, which secured for him in a high degree their affection and esteem.

Diminutive in stature, but full of enthusiasm, activity and lively humor, he seemed to his pupils like a friendly gnome, conducting them through his underground realm, and explain-



Mit begründeter Hochachtung
Ihr
F. Gätzschmann

ing its mysteries with merry delight. After they had left the Academy, he followed them with interest throughout the world, corresponding with them, advising and inquiring concerning their varied work, and manifesting an intelligent sympathy with the progress of the art in which he had first instructed them. I treasure many letters in his queer handwriting, not easy to decipher, but always worth the trouble it cost me to make them out; and I do not doubt that others of his pupils can say the same.

His lectures were characterized by abundant erudition, under the control of practical judgment. He was thoroughly familiar with the actual work, as well as the literature, of mining; and the tendency of his instruction was to make mining engineers, rather than learned commentators and critics.

These features characterized to a large extent the whole circle of great teachers who gave to Freiberg its influence and fame, and of whom Gaetzschmann was the last survivor. Such men as Weisbach, Cotta, Breithaupt, Scheerer, and their associates could not have been easily matched for special ability; but the extraordinary reputation which the Freiberg Academy acquired under their hands was due also to a combination of equally unusual conditions. In the first place, they were all in touch with the practical art of which they taught the theoretical elements; and their scientific activity, by word or pen, was directed to the promotion of that art. Hence the text-books which they produced were mightily helpful and universally welcome to practitioners as well as theorists. Again, they wrought in an almost virgin field, and without the numerous well-equipped competitors who now (largely as a result of their labors) occupy it. The mines and metallurgical works of the *Erzgebirge* led the world in their perfection of method and administration, and reinforced with impressive object-lessons the instruction of the Academy. Moreover, the intimate relations between the professors and their pupils permitted an amount of direct personal training not often possible in such institutions; and the result was a thoroughness of education not always guaranteed by the degree of a technical school. To these causes may be ascribed the exceptional effect which Freiberg has produced upon the arts of mining and metallurgy in all civilized countries. There are still great teachers and investi-

gators in her faculty—such men as Richter and Ledebur, or the lamented Stelzner (recently removed by death), have had no superiors in the present generation. But they do not stand in such conditions as did their illustrious predecessors; and they cannot reproduce the unique glory to which those conditions essentially contributed.

To that brilliant chapter in the history of mining and its kindred arts, the departure of Gaetzschmann, the last survivor of the old régime, writes "Finis;" and we close the book.

Biographical Notice of Franz Pošepný.

BY R. W. RAYMOND, NEW YORK CITY.

(Atlanta Meeting, October, 1895.)

On the 27th day of March last, the day on which the Florida sessions of the last meeting of the Institute began in Ocala, occurred the death of one of its most distinguished honorary members, Bergrath Franz Posepny, of Vienna, formerly professor of the science of ore-deposits in the mining school at Przibram, Bohemia. Prof. Posepny had greatly increased his fame among American mining engineers, besides laying the Institute under special obligations of gratitude, by the elaborate, brilliant and suggestive treatise on "The Genesis of Ore-Deposits," which he contributed to the International meeting, held at Chicago, in 1893. This essay, constituting the first complete publication of the substance of his course of lectures at Przibram, enriched and perfected by the matured results of his investigation and reflection down to the date of its completion, was a free gift of almost unprecedented value to the society which had distinguished him by honorary membership. In estimating the generosity of the author, it must be borne in mind that the copyright of such a work, the fruit of years of study and of practice as an instructor, is of no little value to a European professor, and constitutes one of the legitimate rewards of his (otherwise not highly-paid) labor.

Moreover, Prof. Posepny performed under peculiar difficulties

Mit herzlichem Glück auf, Ihr
Ergebenster
F. Pošepný

his promise to contribute this treatise. Apart from his failing health, an accidental fall had so crippled his hand that he was for months unable to write; and the whole of the voluminous German manuscript had to be dictated to his wife, in whose exquisitely clear and beautiful handwriting it came to me for translation and publication. That interesting labor, willingly performed, was greatly lessened by this circumstance; and I did not hesitate to confess to Prof. Posepny that my personal regret for his accident was considerably mitigated by the indirect gain thus occasioned to his translator.

I trust that I do not transgress propriety by saying in this place a few words concerning Madame Clotilde Posepná, who accompanied her distinguished husband in his visit to the United States in 1876 (as on so many of his other journeys and expeditions), and with whom so many members of the Institute had the pleasure of becoming acquainted at that time. With the exception, perhaps, of Sir Charles and Lady Lyell, I can recall to mind no other husband and wife so highly accomplished, so thoroughly united and so mutually complementary in scientific work. In the matter of languages, for instance, I remember hearing one of them say that, drawing a meridian through eastern Europe, they had divided the map between them; he assuming for his province the tongues east of that line, while she took care of those to the west, including all the European languages and literatures that we commonly regard as required for linguistic accomplishment. German, of course, was common ground to both. The inestimable value of such a colleague to Prof. Posepny is indicated abundantly in his treatise, already mentioned, which exhibits, on the one hand, the results of much original investigation in eastern Europe, and, on the other hand, a wide acquaintance with the technical literature of western nations. That treatise aroused so much interest among mining engineers in this country, and gave rise to so much suggestive discussion, that a separate volume, containing the original paper, the criticisms which it elicited, and Prof. Posepny's reply thereto, carefully indexed for convenient consultation, has been issued by the Institute, to accommodate instructors and students. It was just as this edition was leaving the press that I received the news of Prof. Posepny's death; and in view of the part which his wife had

taken in his service to the Institute and to science, I inserted at the beginning of the book these words, which I here repeat, not doubting that they will be heartily adopted by every one who shall read them:

TO

MADAME CLOTILDE POŠEPNÁ,

WIFE, COMRADE AND COLLEAGUE

OF THE DISTINGUISHED AND LAMENTED

AUTHOR OF THIS TREATISE,

THE PRESENT VOLUME IS INSCRIBED

IN WITNESS OF GRATITUDE FOR HER CO-OPERATION,

AND SYMPATHY WITH HER BEREAVEMENT.

In attempting to sketch the career of Franz Posepny, I shall make free use of the appreciative obituary notice written by his friend and colleague, Oberbergrath Ritter C. von Ernst, one of the editors of the *Oest. Zeitsch. für Berg- und Hüttenwesen*, and published in that journal April 27, 1895.

Born March 30, 1836, at Starkenbach, in Bohemia, Posepny, after preliminary courses in various Bohemian schools, entered, in 1852, the Polytechnic School at Prague, with the special purpose of pursuing the natural sciences, for which he had a native inclination. In addition to the prescribed curriculum, he zealously frequented the lectures and practical exercises in botany, mineralogy, geology, palæontology, chemistry, technology, metallurgy, etc. In order to utilize in the department of mining his knowledge of geology, he went, in 1857, to the mining school at Przibram, where he was specially interested in the lectures of Director Grimm on the science of ore-deposits. It was from Grimm (says his German biographer, on the authority of Posepny's own notes) that he heard for the first time the view that ore-deposits are characteristically confined to decomposed rocks—a doctrine which guided and influenced him for many years after. After finishing his mining course, he entered (1859) the State service, and was first assigned, without pay, to the government bureau at Nagybánya, and thence (1860), at a salary of less than 50 cents a day, to Ohlaposbánya, in Transylvania. This region, with its complicated mine-workings and vein-phenomena, was peculiarly interesting

and stimulating to an ardent young mining engineer and investigator; but he was condemned to the prosaic drudgery of auditing the old accounts of mines which had been destroyed in the rebellion of 1848, and his superior official discouraged his studies underground, telling him that he had "much more important things to attend to than going down into old mines, which could show him only rubbish and dirt." He was obliged, therefore, to pursue his favorite studies in secret until a more favorable position was assigned to him as the director (at about 60 cents per day!) of certain explorations for lignite in the district of Kovar. Here he distinguished himself by the execution of a topographical and geological map of the district, determining, on palæontological evidence, the age of the coal-deposits. This led to a recognition of his peculiar qualifications for the study of problems in economic geology; and in 1862 he was designated (at the increased salary of 75 cents per day!) to make an investigation of the ore-deposits and almost abandoned mines of Rodna, in Transylvania. This work, in which he at first received assistance from the *Geologische Reichsanstalt* at Vienna, was subsequently somewhat peremptorily and prematurely terminated, and, late in 1865, Posepny was ordered to make a similar study of the gold-mines of Verespatak. This occupied him until 1869, when he was recalled to Vienna, and directed to examine and report upon the mines of Raibl, in Carinthia. This work consumed a good deal of time, and the authorities were, perhaps, inconsiderate in their repeated demands for a hasty completion of it. Posepny was still, after 11 years of service, only an "expectant," without title and with scanty pay; and in justifiable dissatisfaction with this treatment, he accepted, in 1870, the offer of an independent position—specially created for him—as economic geologist for Hungary, with a salary and allowances amounting to something like $1000 per annum. This he occupied for two years, executing during that period many investigations of value to the Hungarian mining industry. In 1872 he returned to Raibl; finished and presented, in 1873, his official report on that district, and then went back to Hungary, to continue his study of the Schemnitz region. But by this time his services were required in a wider field; and he resigned his position in Hungary to accept that of Vice-Secretary in the Royal-Imperial

Ministry of Agriculture (including mining) of Austria. In this capacity, from 1873 to 1879, he carried out in Tyrol and Salzburg a series of investigations (published in the first volume of his *Archiv für Praktische Geologie*), and also made journeys to various countries, including an extended tour in the United States.

But he was not satisfied with this achievement of official position and its sphere of usefulness. His conviction of the importance to the mining industry of the scientific study of mineral deposits had been expressed incessantly in publications, urging the introduction of special lectures on this subject in mining schools; and in 1879, the ministry with which he was connected succeeding in obtaining from the Emperor authority to establish, at the academies of Leoben and Przibram, separate chairs devoted to that department. The professorship at Przibram, together with the title of Bergrath, was given to Posepny, and occupied by him until, in 1888, he retired from public service, receiving in recognition of his merit the order of the Iron Crown.

In some respects, his labors at Przibram were the most fruitful of his life. Besides discharging the duties of the class-room, which served, no doubt, to consolidate and systematize the knowledge gathered in practice, he added to that knowledge by a diligent and minute study of the geology and vein-relations of the extensive and productive Przibram mines. This really great investigation was carried through by Professor Posepny with wonderful persistency, at great personal expense, and without assistance. Its results are to be published in the second volume of his *Archiv für Praktische Geologie*, which was in press at the time of his death.

After resigning his professorship and retiring from active service, he established himself, with his inseparable helpmate, in a pleasant cottage home in the suburbs of Vienna, where he devoted himself more exclusively than ever to his favorite studies, making journeys of observation to Transylvania, Germany, Switzerland, the Ural, France, England, Sweden, Norway, Italy, Sardinia, and finally, in the spring of 1894, to Greece and the Orient, as far as Jerusalem. His principal attention in these journeys was given to the geology and the mining (present, historical or pre-historical) of the countries he visited.

That he could appreciate, however, other sentiments and associations, I have touching proof in a note which he sent me from Jerusalem, enclosing a leaf plucked on the Mount of Olives. It should be mentioned also that, in addition to his main specialty, he was an enthusiastic student, and no mean authority, in anthropology and numismatics.

During this closing period of his intensely active life, his industry might fairly be called desperate; for the increase of a long-standing pulmonary weakness, to which in these latter years a disease of the heart was added, produced in him the abiding conviction, not only that his days were numbered, but that they might at any moment suddenly end. What he accomplished with failing strength and under such a depressing consciousness, is truly amazing. Yet, in his letters to me, he never alluded to the shadow of such an apprehension; and I did not dream that his magnificent contribution to the Institute was the bequest of a dying man, and the last important work of his life. I take the liberty of translating portions of a private letter from his wife, which, although not intended for publication, are calculated to give, better than words of mine could do, a pathetic and inspiring picture of his heroic devotion:

"Although for many months I had necessarily foreseen the sad termination of his sufferings, I could not help clinging to occasional momentary gleams of hope; and the end seemed, after all, awfully sudden.

"Only with the utmost exertion did we two succeed in so far completing the proof-reading of the second volume of the *Archiv*, that nothing will now prevent its early publication.

"With the kind assistance promised by his professional colleagues, I may also hope to bring out, in a year or two, a third volume. It is a purpose dear to me to publish all that he left behind. Much will, of course, appear in fragmentary form, but it will at least stimulate thought and discussion.

"It is almost incredible how hard he worked, giving himself in later years no rest, because he continually looked for death. Outwardly he appeared so full of life and pleasure in life (*so lebensfroh*), and seemed to be in perfect health. But I knew better; and he himself used to be annoyed when people spoke of his good looks, for, as he said, he was always 'only a handsomely turfed grave!'"

I am unable to give at this time a complete list—still less a critical account—of the published reports and treatises of Prof. Posepny, between one and two hundred in number. This will be done, I understand, in the introduction to the second volume of his *Archiv*, now in press. Nevertheless, I may venture to

express some general reflections concerning his career and his position in scientific literature.

1. Even from the bare outline of his life which I have given, it is evident that he trod no easy path to eminence and fame. For many years he was utilized without being adequately appreciated; ordered from place to place; scantily paid and arbitrarily overruled; his far-reaching plans thwarted by short-sighted officialism, intent upon more immediate practical results. For this the government bureaus are not necessarily to be blamed. Posepny was, heart and soul, not a government official, but the lover and slave of science. And governments do not exist for the promotion of science. The utmost which they can legitimately do in that direction is to assist the progress of science on grounds of political economy; that is, as an element in the industrial prosperity of the commonwealth, and an incident of the intelligent administration of its resources. European states have gone further in theory than our own Federal government (though few have been so loosely liberal in practice) in the range of application given to this principle. But, under any government, immediate administrative necessities may often take precedence of purely scientific investigations, and the subordinates of a bureau may be commanded to devote themselves to barren routine when they would rather be "exploring the unknown."

2. Moreover, not everybody who burns with ambition to distinguish himself by increasing the sum of permanently valuable human knowledge should, on that account, be enabled, either by public or by private aid, to pursue his supposed mission at the expense of other people. Some peculiar fitness must first be demonstrated; and, on the whole, there is perhaps no better test than that of patient and obedient service, even under unwelcome restraint. The man who, like Posepny, in spite of, and in addition to, his routine duties, continues with ardor his scientific investigations, is the best man to be subsequently intrusted with such higher work.

3. But this is not all. The best training, even for a specialty, does not consist in simply encouraging the inclination of genius in one direction. We hear a good deal about education as being ideally, as it is etymologically, the "drawing-out" of what is already in the pupil. This is true enough, if we add

that the best work of education is the drawing-out of faculties which the pupil does not know or believe to be in him, and that its least important function is the assistance of those dominant powers and purposes which need little help. It is often in the branch for which the schoolboy shows no taste or capacity that he should be most rigorously drilled, not merely for the moral, but also for the mental, discipline thus secured. And there is nothing that contributes more potently to success in the larger school of life than the subjection of young men to work which they do not like, and in the knowledge of which they are, consequently, deficient. I say "consequently," but the consequence may be often the cause. The dormant capacity once developed by practice, many a man ends by liking a work which he understands, who began by disliking it because he did not understand it.

4. In the case of Posepny, I am not at all sure that the disappointment and drudgery of his early career were not the best things that could have happened to him. Incidentally, they gave him a much wider experience than he would have obtained by rapid promotion—which might have made of him, either a conservative official, calmly contemptuous of youthful ambitions, or a library-theorist, learnedly discoursing of nature at second-hand; of both of which classes the world has enough already. They are useful in their way; but it would have been a pity to waste Posepny, in order to increase either.

The result, in his case, of the irksome discipline of fiery, unconquerable genius, was to reinforce the knowledge of literature and theory with an extensive and intimate direct knowledge of nature, and, above all, to make the chemist and geologist also a practical miner and mining engineer. The latter circumstance adds exceptional and characteristic weight to his scientific generalizations. I may add, that in my judgment, the nature of his early labors not improbably bred or deepened in him that sense of the vital importance to science of the minute observation, and purely "objective" description, of single groups of phenomena, which is so prominent in all his writings. In accordance with it, his works are mainly detailed accounts and discussions of single mining districts. In other words, he continued to the end the method of investigation which was forced upon him in the beginning by superior

authority. The difference between such monographs, produced by the patient labor of months in each locality, and the sketchy results of hasty visits by expert tourists, such as constitute much of the literature of this class, requires no comment.

5. I have emphasized at some length this feature of Posepny's work, because I think it carries an important lesson for American mining engineers and geologists. We are making rapid progress in science; but we do it in a tumultuous and irregular fashion, accumulating a goodly stock of untrustworthy data and of premature theories as we go. Our young investigators are often in a hurry to promulgate generalizations; and on the other hand, our practicing mining engineers are often too busy to observe and record facts. The two classes could aid each other more than they do; and especially those who are confined by their duties to one locality might learn from the example of Posepny that the thorough study of one locality is the most valuable contribution that can be made to general science. On the other hand, the authors of theories may profitably note that Posepny himself, as the result of wider observation, was obliged to change the views he had expressed, under the influence of preconceived impressions, in early years.

6. In my brief preface to the separate edition of "The Genesis of Ore-Deposits," issued by the Institute, I have used the following language, which I here repeat, as an introduction to some further observations upon Posepny's work.

"The views of Cotta and his associates, sometimes called for convenience 'the Freiberg school,' dominated for a generation the current theories and classifications of mining engineers. This is particularly true of the United States, where the excellent translation of Cotta's text-book by Prof. Frederick Prime, Jr., one of his pupils, was for many years the controlling, and indeed the only easily available, authority on this subject in the English language.

"As a personal friend, diligent student and hearty admirer of Bernhard Cotta, and no less as professional critic of his views, I feel myself bound to say that his theories, as stated more than thirty years ago, are still, to a surprising degree, valid and comprehensive. There is scarcely a single modern modification of them for which he did not, with intuitive prescience, leave a

place. On the other hand, it is a fair criticism of the whole 'Freiberg school,' that it gave too much prominence and attributed too much typical importance to fissure-veins of the class represented in the *Erzgebirge.* Such writers as Groddeck and Grimm have undoubtedly aided to modify this disproportionate emphasis. But it has not ceased to influence the conceptions entertained by miners, and even by legislators, as the United States mining law (evidently based on the 'true fissure-vein' as a general type) abundantly demonstrates."

Of the two authorities named in the above extract, as aiding to modify the views of the "Freiberg school," Bergrath Dr. Albrecht von Groddeck, whose treatise appeared in 1879, was the director of the Prussian Mining Academy at Clausthal. His treatment of the science of ore-deposits was chiefly characterized by the recognition of numerous "types," and the citation of leading examples under each type. Oberbergrath Johann Grimm, whose treatise appeared in 1869, was director of the Austrian Mining Academy at Przibram, in Bohemia; and it was in Grimm's lecture-room, from 1857 to 1859, that Posepny received his first working-theory of the nature and origin of mineral deposits. I must confess that I cannot find in Grimm's book, published ten years later, the sweeping generalization to which, on Posepny's authority, Ritter von Ernst (as quoted by me above) alludes; and I am led to suppose that the veteran instructor had seen cause, before 1869, to modify his views. However that may be, it was as a disciple of Grimm that Posepny began his work; and it was only after years of patient study of facts in the field, that he promulgated any comprehensive system of his own.

7. That system, his matured statement of which is found in "The Genesis of Ore-Deposits," cannot be said to involve any appeal to newly-discovered causes, or any denial of accepted principles in geology. The same is true of all systems proposed since the exclusive agency of plutonic action on the one hand, or of aqueous action on the other, was recognized as untenable. They have all been simple attempts to classify the observed facts for fruitful study, and to estimate the relative importance of the several natural agencies which were universally recognized as factors. For the purpose of classification, the chief distinctive characters have always been: (1) The time-

relations of a mineral deposit, as formed simultaneously with the enclosing rock, or as a regular member of a series of rocks, or as a later segregation or intrusion; (2) its form; (3) the manner and agencies of its origin; and (4) its contents. Of these characters, sometimes one and sometimes another has been treated as the primary distinction. Gold-, silver-, lead-, and copper-mines, etc., may have been the leading classes in a system designed for convenient use in practice; veins, stockworks, and impregnations, may have been separated as groups of independent significance in another practical system; original deposits may have been combined with deposits of subsequent formation, if both were supposed to have originated through the same processes, etc. For the purposes of science, it will probably be admitted that a genetic classification is to be preferred; and such a classification Posepny proposed. That it was not final or complete he acknowledged, not only expressly in words, but tacitly by his preliminary division of minerals as "idiogenous" and "xenogenous," and the practical confinement of his genetic classification to the latter. It is, of course, plain that the idiogenous minerals must likewise have had a genesis, and that a complete genetic classification would include them, not as a separate primary group, but as parts of other groups, determined by the conditions and agencies of their origin. Posepny's system, beginning as it does with the rocks already formed, and ignoring their prior genetic history, is, to that extent, an avowed compromise. But it is on that basis to be judged, and not by comparison with something more ambitious and comprehensive, at which the author did not pretend to aim. My views on this subject have been sufficiently set forth elsewhere; and the position of Posepny has been so clearly and fully stated by himself as to render further exposition needless.

8. In fact, the present state of the science of mineral deposits is such as to render any man's system of classification a matter of subordinate pedagogic importance. The declared purpose of Professor Posepny, in the presentation to the Institute of what he at first entitled "Subjective Views of the Origin of Ore-Deposits,"* was to invite criticism and discussion.

* See my remarks, *Trans.* xxiv., 980, and in the separate Posepny vol., page 233.

This purpose was unquestionably realized in a discussion (not yet ended) of great interest and value. And the remarkably stimulating effect of that treatise seems to me typical of the chief permanent effect of the author's whole work, in the field in which he became pre-eminent. I believe it will be the verdict of his successors, as of his contemporaries:

a. That he furnished an example of unselfish and unqualified devotion to science, which will be an inspiration forever.

b. That he contributed to science, in his special department, an immense amount of careful and accurate fundamental work, which can be confidently relied upon as trustworthy material for future study, being guaranteed, not only by his eminence in general science, but also by his familiarity, as a mining engineer, with operations and observations underground.

c. That, on the basis of his wide observation, coupled with his extensive knowledge of technical literature, he exerted a potent influence in promoting the scientific study of ore-deposits and in correcting extreme theories and tendencies which have tended to bias and distort that science.

9. In the last proposition, I have in mind more particularly the controversy which Posepny and his friend, the late Prof. Stelzner, of Freiberg, waged against the lateral-secretion theory of Prof. Sandberger. In this debate Posepny no doubt assumed to some extent the attitude of a partisan; and, perhaps, in some respects, his controversial utterances may have gone beyond a judicial impartiality. This has been pointed out more than once by his critics, and particularly in the discussion of his recent treatise in the *Transactions* of the Institute. As I have elsewhere declared, I think he was right in his general view and argument, and I will here do no more than call attention to the circumstance that those of his statements which have been seriously contested by American authorities were mainly based upon the publications of others, not upon his own observation. Many such publications are affected with "subjective" opinions; many of them are unaccompanied with accurate drawings; and many of them lack precision in description, and are, therefore, liable to misinterpretation. It is no wonder that in single cases Posepny may have mistaken the intended meaning of an author or accepted too hastily an assertion too hastily made. But it must be confessed that, as a

whole, his survey of the literature of his subject was singularly comprehensive, intelligent and fair.

10. As I have observed already, Posepny's final work was not offered as the last word of science in that field. We now know, what he knew when he wrote it, that it was *his* last word—the utterance of one who was about to turn over to others the results of a life-labor still incomplete, and surpassing in fruitful suggestion even its illustrious record of accomplished achievement. The loss of such a man at any time is deplorable; but doubly so when he departs in the prime of years, just prepared for the ripest and richest harvest of all his planting. Posepny's views will still incite and reward discussion; but we shall sorely miss the ablest of expositors and critics in Posepny himself.

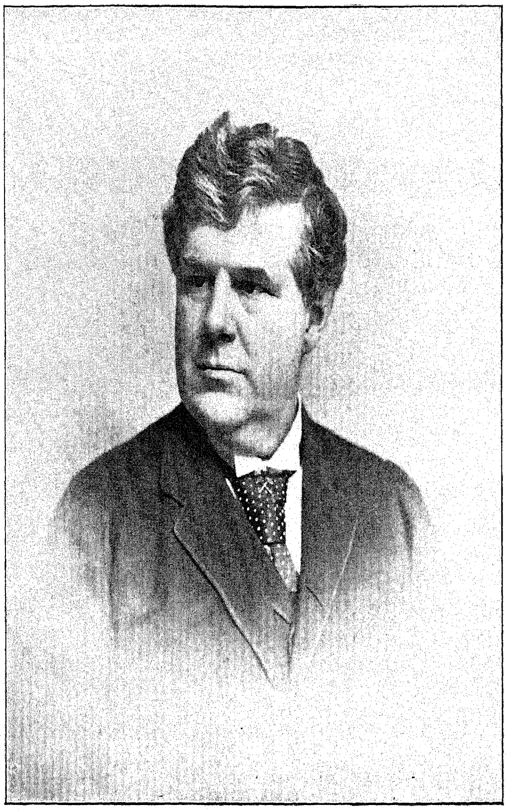
It has been my endeavor in the foregoing sketch to preserve the standpoint of disinterested justice; but I cannot deny that, while I have been thus coldly analyzing and estimating the scientific leader, there has been constantly present with me a vision of the splendid presence of my own dear friend. I never saw him but once—at the time of his visit to this country in 1876. But that meeting confirmed the personal attraction already exercised upon me by his works; and our subsequent intercourse by correspondence made the charm perpetual and indissoluble. A few such friends I may still count in foreign lands, unseen, yet ever present; and it sometimes seems to me that these relations of mind and heart, which defy separation in space, are the best types of the relation which defies death also. At all events, I find that he and I, who could be together, though confined to the Old and the New World respectively, are not less mutually near, now that one of us has entered the World which lies so close to both.

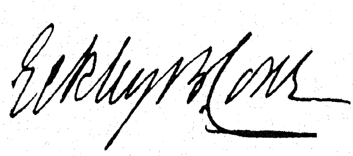
Biographical Notice of Eckley B. Coxe.

BY R. W. RAYMOND, NEW YORK CITY.

(Atlanta Meeting, October, 1895.)

THE relations of Eckley B. Coxe to the Institute, as one of its founders and early Presidents, and, throughout its history, an enthusiastic supporter and a valued contributor, would alone





justify the publication in its *Transactions* of an appreciative notice of his life, and of our loss by his death; and this consideration is reinforced by the universal esteem and love commanded by his personal character, throughout a large circle, embracing all who knew him, and especially his fellow-members in this society. But the connection of his family, and of his own activity, with the history of this country, and with the industrial progress of our generation, is a consideration of yet wider significance; and I offer, therefore, no apology for giving to this imperfect survey of his career a wider range than his connection with the Institute would require, or my own knowledge will enable me adequately to cover. I can only say, that the deficiencies of my sketch will be those of omission. I believe that the positive statements (which I have taken pains to verify) may be relied upon as correct.

Of the ancestors of Eckley B. Coxe, the earliest of whom I have found a record was Daniel Coxe, of Westminster, London, to whom, in 1648, a coat-of-arms was granted by Parliament. His son, who died in 1686, and his grandson, born about 1640, both bore the same name; and the latter, Dr. Daniel Coxe, who lived in and near London to the good old age of ninety, was the first of the family to establish relations with the New World of the West. As he received in 1669 the degree of Doctor of Medicine from Cambridge University, was made in 1683 an Honorary Fellow of the Royal College of Physicians and Surgeons, was one of the physicians of Charles II. and the medical attendant of Queen Anne, and is recorded as among the earliest scientific experimenters with certain drugs upon animals, it is evident that he enjoyed a social and professional success likely to command the rewards of wealth. But that he was interested in matters beyond his immediate profession is shown by the fact that, somewhere between 1692 and 1698, he purchased the patent of the Province of "Carolana," which had been originally granted by Charles I. to his Attorney-General, Sir Robert Heath. This patent covered (with some reservations) the territory extending from the Atlantic to the Pacific, between the 31st and 36th parallels of North latitude. The title had doubtless been somewhat impaired—legally, perhaps, by neglect of the previous holders to explore and colonize with diligence; practically, through actual occupation of parts of the territory

by others. Dr. Coxe made energetic attempts to revive it, at least so far as unoccupied portions were concerned. An expedition sent by him from Charleston, in 1698, reached the Mississippi near the place where De Soto had discovered it, 157 years before; and two vessels, fitted out by him in the same year, are said to have been the first to find the mouth of that river, and to have sailed up it for 100 miles. It does not appear that actual possession or colonization was maintained; but evidently enough was done to preserve some rights under the patent. Dr. Coxe was fifty years old when he organized the expedition above named; and although he lived forty years longer, it may be presumed that his energy, or his means, diminished with increasing age. Yet that, thirty years after his death, his title to the indefinite, but immense, province of "Carolana" was still deemed to possess some value, may be inferred from the fact that his grandchildren, upon its surrender to the Crown in 1769, received as compensation a grant of 100,000 acres in the Colony of New York.

Meanwhile, in 1684 and 1686, Dr. Coxe acquired lands in East and West Jersey, by purchase from the Byllinge family of the territories and rights previously granted to them by the Duke of York, and became Governor of West Jersey. In this capacity he did much to develop the industries of the province by establishing whale-, cod-, bass- and sturgeon-fisheries, the manufacture of salt (from sea-water), the manufacture of pottery at Burlington, the exportation of timber, and the West India trade.*

Colonel Daniel Coxe, a son of Dr. Coxe, born in 1673, came to America in 1700, and resided in New Jersey until his death in 1739. He took a prominent part in the affairs of the Colony, being at different times a member of the Royal Council, speaker of the Assembly, and Judge of the Supreme Court. He was also, in 1730, Provincial Grand Master of the Free

* For further details of the life of Dr. Coxe and of his son, Colonel Daniel Coxe, see the "Biographical Notice of Dr. Daniel Coxe," by G. D. Scull, of Oxford, England, in the *Pennsylvania Magazine of Hist. and Biography* (published by the Historical Society of Pennsylvania), vol. vii., page 317. There exists also a work by Colonel Coxe, which I have not seen, entitled, *A Description of the English Province of Carolana*. It was issued at London, in 1722, and is noteworthy as containing what was probably the earliest published plan of political union for the American colonies.

Masons of the Middle Colonies, and thus, perhaps, the earliest Masonic Grand Master in North America. In 1722, he published in London the *Description of the English Province of Carolana* referred to in a preceding foot-note.

His son, William Coxe, a merchant of Burlington, N. J., and Philadelphia, Pa., born in 1723 and dying in 1801, was the father of Tench Coxe, whose name is honorably associated with the political and industrial history of the United States. Tench Coxe was a delegate to the Annapolis convention of 1786, and to the Continental Congress in 1788; Assistant Secretary of the Treasury under Alexander Hamilton, in 1784; Commissioner of the Revenue in 1792; and Purveyor of Public Supplies from 1803 to 1812. He is especially entitled to remembrance for his labors in behalf of American manufactures and his statistical contributions to political economy. He was the first to attempt the introduction of the Arkwright loom into the United States, and the first to urge the people of the South to raise cotton.*

Alexander Hamilton said of him:

> "In examining American writers on the subject, I find no individual who commenced so early, and who continued with such unswerving perseverance in the particular promotion of the growth of cotton as the only redundant staple which this country could produce: in the commencement and forwarding of the cotton manufactures under really disadvantageous and great embarrassments, I find no one appearing at the head and front of these measures equal to Tench Coxe."†

At various times between 1787 and 1794, Tench Coxe wrote a series of papers, which were collected and published at Philadelphia, in 1794, under the title, *A View of the United States of America.* In these remarkable essays, most of which were written during the period of the Confederation, he advanced the most enlightened views concerning the development of American manufactures and commerce. That he was aware both of the existence of coal in middle as well as western Pennsylvania, and of its prospective industrial importance, appears from the following passage, which I quote from the work above cited (page 70):

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* See, for a mention of Tench Coxe as the highest authority of his time on American manufactures, etc., the chapter by David A. Wells on the Progress of Manufactures, in *The First Century of the Republic*, page 160.

† See Hamilton's *Report on Manufactures*, 1791; also the *Memoirs of Samuel Slater*.

"So many of the necessary and convenient arts and trades depend upon the plenty and cheapness of fuel that it appears proper to take notice of this article. Till the Revolution our dependence was almost entirely upon wood-fuel, of which, in the most populous places, we have still a great abundance, and in all interior situations immense quantities; but the increase of manufactures has occasioned us to turn our attention to coal. Of this useful fossil Providence has given us very great quantities in our middle and western country. The vicinity of Wyoming, on the Susquehanna, is one bed of coal of the open-burning kind and of the most intense heat. On the head-waters of the Schuylkill and Lehigh are some considerable bodies. At the head of the Western Branch of the Susquehanna is a most extensive body, which stretches over the country southwesterly, so as to be found in the greatest plenty at Pittsburgh, where the Allegheny and Youghiogheny unite and form the Ohio. It has been lately discovered on the waters of the Nescopeck. All our coal has hitherto been accidentally found on the surface of the earth or discovered in the digging of common cellars or wells; so that when our wood-fuel shall become scarce, and the European methods of boring shall be skillfully pursued, there can be no doubt of our finding it in many other places."

The author of this prophetic utterance had the courage of his opinions. In 1787 and 1793, in partnership with others, he acquired many tracts of land in Pennsylvania.* The valuable coal-lands now owned by his descendents are part of these purchases, and the collieries now operated on the Tench Coxe estate are on the waters of the Nescopeck, where he pointed out the existence of coal more than a hundred years ago.

But a generation passed before these undeveloped resources could be effectively utilized; and this period of difficulty and delay includes the active career of Charles Sidney Coxe, the son of Tench Coxe and the executor of his estate. He was a lawyer, and became District Attorney of Philadelphia and Judge of the District Court; but, thoroughly penetrated and possessed by his father's views, he made it the chief business of his life to keep together the large body of coal-lands inherited by himself and his brothers and sisters. The undivided estate was left in his sole charge. The land was entirely unproductive; the amount of annual taxes was large; squatters and

* These lands, purchased from the State of Pennsylvania, comprised about 172,000 acres, and were situated in what are now the counties of Somerset, Indiana, Jefferson, Pike, Wayne, Susquehanna, Bradford, Northampton, Centre, Union, Columbia, Schuylkill, Carbon, Luzerne and Northumberland, of which the last four contained, for the most part, the coal-lands preserved by the descendants of Tench Coxe. The tracts now belonging to the Coxe estate, and including valuable coal-basins (though not everywhere underlain by coal), amount to about 26,000 acres, and are, with the exception of small portions still under outstanding leases to other parties, leased to the Cross Creek Coal Co.

timber-thieves had to be kept off; and many adverse titles, arising from tax-sales, conflicting surveys, etc., had to be settled by compromise or litigation. His policy was to make every sacrifice to retain the coal-land, and to secure means for this purpose by selling outlying farms and timber-tracts. He knew every corner and line of the property, having personally traced them all on the ground; and, possessing a good knowledge of the geology of the region, he spent every summer in determining, by shafts and borings, the boundaries of the coal-basins. Born in 1791, he died in 1879; and at the time of his death all conflicts of title had been practically quieted, a number of coal-leases had been made, the properties had begun to pay satisfactory royalties, and his sons had been thoroughly trained to administer the valuable estate and the immense business which his wisdom and energy had preserved.

Judge Coxe had seven children, five sons and two daughters. The eldest son, Brinton Coxe, was a lawyer and a writer of eminence on constitutional law. The other four sons, of whom Eckley was the second in age, became associated in the coal business. One of them died in 1873; the other two, Alexander B. and Henry B. Coxe (and one sister), still survive. It was, however, Eckley B. Coxe upon whom, even before his father's death, and afterwards as executor of the estate, that the chief responsibility of technical management was laid.

Eckley Brinton Coxe was born at Philadelphia, June 4, 1839. In 1854 he entered the University of Pennsylvania, whence he was graduated in 1858, at the age of 19. While a student he was assistant in the laboratory of John F. Frazer, Professor of Chemistry and Physics, and after his graduation he took an additional scientific course at the university, and studied also French *and book-keeping*. The latter item is significant of his own practical wisdom, or of his father's. A knowledge of accounts is as essential to mining engineers and managers as any branch of physical science or professional training. It is evident that the young student, conscious of the work to which he was called by the traditions and circumstances of his family, prepared himself for it from the beginning. Much preparation, indeed, he received almost unconsciously; for during his school and college years all his summers were spent in the coal-regions, where the family sojourned every year. Here he

accompanied the surveyors and explorers, frequented the small foundries and machine-shops, the mines and breakers of the region, and became at an early age familiar with the practical details of mining, surveying and machinery. Such a boyish knowledge of practice was unquestionably of great value in his subsequent study of theory. But perhaps still more important was the effect of his association as a boy with the working miners themselves. This gave him a sympathetic knowledge of their circumstances, habits and modes of life and thought which he never lost, and which was of the greatest service to him as an employer.

In 1859 he was engaged under Mr. Benjamin Smith Lyman (still an active member of the Institute) upon a topographical survey of part of the Tench Coxe estate.

In 1860 he went to Paris and spent two years as a student in the *École des Mines*. Of the esteem in which he was there held, and of the relations which he sustained to the end of his life with that institution, I can give no better evidence than the following letters, which I have translated from the originals, kindly loaned me by Mrs. Coxe, to whom they were addressed:

THE FRENCH REPUBLIC, MINISTRY OF PUBLIC WORKS.
PARIS, June 26, 1895.

MADAME: The Director of the *École Nationale des Mines*, at Paris, has just informed me of the decease of M. Eckley B. Coxe, your husband, a former student in that school.

Mr. Coxe had preserved a grateful remembrance of the instruction he had received in that great scientific establishment, and had given numerous proofs thereof by the donation, on different occasions, of important and remarkable collections. To these he had very recently added the gift of an extensive series of American specimens; and it was at the moment when I was about to thank him, in the name of the *École des Mines*, that I received the sad tidings of his death.

I venture, Madame, to express to you my sympathy in this sorrowful event, and to assure you that the *École Nationale des Mines* will preserve the memory of the presence of Mr. Coxe in its class-rooms, and of the generous liberality with which he subsequently enriched its collections.

Receive, Madame, the assurance of my respect,

The Minister of Public Works.
DUPUY-DUTEMPS.

MINISTRY OF PUBLIC WORKS,
ÉCOLE NATIONALE SUPÉRIEURE DES MINES,
PARIS, June 28, 1895.

MADAME: It is with profound grief that I learn from the newspapers the death of the eminent, excellent, and deeply-lamented Eckley B. Coxe. I have the

honor to transmit to you a letter in which the Minister of Public Works himself expresses the profound regrets of the French government, and its gratitude for the affectionate remembrance which Mr. Coxe had always preserved for the *École Nationale Supérieure des Mines.* He was one of its most distinguished foreign pupils, and he has overwhelmed it with repeated and most precious contributions to its collections.

It is certainly permissible, Madame, to inform you that on the 27th of May I addressed to the Minister of Public Works a letter, requesting him to recommend to his colleague, the Minister of Foreign Affairs, a decree, conferring upon Mr. Coxe the decoration of Chevalier of the National Order of the Legion of Honor. His decease, so suddenly intervening, has prevented decision upon this recommendation. I expected that it would be favorably considered, but can now only express to you my personal opinion, the proceeding having been so sadly cut short before any examination on the part of the advisers of the government.

I pray you, Madame, to accept the sorrowful expression of my respect and sympathy.

HATON,
Inspector-General of Mines, Member of the Insitute,
Director of the *École Nationale Supérieure des Mines.*

ÉCOLE NATIONALE SUPÉRIEURE DES MINES,
OFFICE OF THE INSPECTOR,
PARIS, June 28, 1895.

MADAME: Permit me to express to you the profound sorrow occasioned to me by the tidings, so unexpected, of the death of M. Eckley B. Coxe, for whom I have cherished for a long time a true affection. We were, for several years, fellow-students at the *École des Mines*, in Paris. A year later, in one of my professional journeys, I found him at Freiberg. I loved him sincerely, and was rejoiced to meet him once more, some years ago, on the occasion of the Paris Exposition of 1889.

Filled with affection for him personally, and with gratitude for the gifts he had made to the *École des Mines*, I supported with all my heart the proposal made by the Director of the *École* that he should be decorated with the cross of the Legion of Honor. We learned with consternation from the newspapers the death of this distinguished engineer.

Accept, Madame, the expression of my entire sympathy and respect.

ADOLPHE CARNOT,
Inspector-General of Mines, Member of the Institute.

After leaving the Paris school, he went to the Mining Academy at Freiberg, in Saxony. Here, as in Paris, he was a zealous student; and he became particularly intimate with Julius Weisbach, the famous professor of mechanics and engineering, whose original investigations and admirable text-book are still unsurpassed in that department. Professor Weisbach authorized him to translate the first part of this great treatise, namely, the volume on *Theoretical Mechanics;* and the ardent young disciple carried out this laborious undertaking, and published

in 1870, after his return to the United States, an octavo volume of 1112 pages as the result.*

He expended not only labor but money in this undertaking; and I doubt if it ever brought him pecuniary profit. But it speedily made him known among students of his profession, and prepared the way for the general recognition of the position which he afterwards held, as the foremost mining engineer of the United States.

It goes without saying, that, upon his return to the United States in 1864, he immediately threw himself with enthusiasm and industry into the work for which he had been trained. A number of coal-leases had been granted upon the Tench Coxe estate (the first bearing date in 1852); and although some of these had terminated or had been forfeited, there were still, in 1865, several large operators occupying as tenants some of the most productive portions of the estate. It became the definite aim of Eckley B. Coxe to consolidate the control of the whole property in the hands of those interested as owners of the land. But he moved towards this end with prudence. In 1865, the firm of Coxe Brothers & Co., organized for the purpose, leased from the estate the Drifton lands, and began the highly successful business career which (in later years under the name of the Cross Creek Coal Company) it has continued ever since. Yet, even as late as 1875, new leases were granted by the estate; and it was not until the operations at Drifton had been thoroughly developed and tested, that the steady absorption under the same control of all the mining on the estate began. By 1886, only 1200 acres of the coal-land of the estate remained outside of the control of the Cross Creek Coal Company, of which Eckley B. Coxe was president; and the area of coal-land united under his management (including territory leased from other owners) was 35,013 acres. The shipment of coal, which had been 26,644 tons during the first year of opera-

* *A Manual of the Mechanics of Engineering and of the Construction of Machines, with an Introduction to the Calculus.* Designed as a Text-Book for technical Schools and Colleges, and for the Use of Engineers, Architects, etc. By Julius Weisbach, Ph.D., Oberbergrath and Professor, etc. In three vols. Vol. I., *Theoretical Mechanics.* Translated from the Fourth Augmented and Improved Edition by Eckley B. Coxe, A.M., Mining Engineer. New York: D. Van Nostrand. 1870.

tions, had reached 1,405,096 tons (not including what was used or sold at the mines) in 1889.*

Then followed a period of skillful and resolute dealing—partly by negotiation, partly by direct contest—with the great question of transportation to market, including the construction of nearly 50 miles of railroad,† by which all the collieries of the company acquired an independent connection with the Pennsylvania, New Jersey Central, Reading and Lehigh Valley systems, and were no longer subject to the dictates of any single transportation-company. This connecting road carried in 1894 more than 2,000,000 tons of anthracite, notwithstanding repeated suspensions of mining caused by the general restriction of production. It is not surprising that, since the Cross Creek Coal Company thus secured its independence, former controversies have ceased, and its relations with all the railroads are exceedingly amicable!

The remarkable business achievement thus outlined may be considered the great work of Eckley B. Coxe's life; nor is its greatness determined by a sordid standard, as though it were merely the selfish consolidation of a vast private fortune. Both

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* Including the coal used and sold at the mines, the amount produced was 1,529,906 tons. The year mentioned was peculiarly favorable, being almost the only recent year during which the collieries of the anthracite region could be operated continuously, without restrictions and stoppages due to commercial causes. The product of coal from the Coxe estate proper was largest, I believe, in 1885, when it amounted to 987,940 or (including coal used and sold at the mines) about a million tons. The collieries now operated by the Cross Creek Coal Company are the Drifton, Eckley, Stockton, Beaver Meadow, Tomhickon, Oneida, Deringer, and Gowen. The last two are served by one breaker; the rest have one breaker each; and at Drifton and Oneida the breakers are of iron, and are provided with electric lights for night-work.

† The Delaware, Susquehanna and Schuylkill Railroad Company was formed in 1890 (Eckley B. Coxe being President, and all the stock belonging to the members of the Cross Creek Coal Company), and the road was opened late in 1891. The road now owns 29 locomotives, 1500 coal-cars (of 60,000 lbs. capacity), and a full equipment for business in other particulars.

I may mention here that two other stock companies were organized, for convenience of management, in connection with this business; Mr. Coxe being the President of both, and the ownership of the stock being the same as in the cases already mentioned. The first is Coxe Brothers & Co., Incorporated, which is essentially a selling agency, with shipping-docks at Perth Amboy, Buffalo, Milwaukee and Chicago. The other is the Coxe Iron Manufacturing Company, to which the machine-shops of the original firm have been transferred, and which, besides the constructions and repairs for the mines and the railroad, executes large outside orders for machinery, etc.

the methods and the motives of this achievement were pure and lofty. The methods were those of open and fair competition; of the honorable performance of contracts; of wise and liberal economy; and of scientific improvements, which reap profit from the resources of nature, not from the sufferings of fellow-men. The motives were higher than those of ordinary so-called philanthropy. The possessor of wealth may be a mere miser, or a mere spendthrift, or a mere annuitant, reaping what he does not sow, and as truly dependent as any pauper upon the bounty of others. Or he may deserve praise for generous gifts, which are to be administered by others. In many instances, no doubt, wealth thus given away is wisely bestowed. But the act is a tacit confession that others can employ, more beneficently than the giver, the power thus resigned. In any case, the ethical merit of the act is measured by the degree in which the actor "gives himself with his gift;" and the highest fulfillment of the New Testament conception of stewardship, as well as of the scientific conception of true philanthropy, is realized when the possessor of the power which wealth confers neither repudiates nor resigns its responsibility, but devotes his life to the administration of it, for the benefit of present and future generations. This is what Eckley B. Coxe did; and it seems to me that his example is well-nigh unique among us, in its combination of the following features:

1. The familiar phenomenon of the success of "self-made" men is, of course, no argument against the value of a thorough education, such as wealth can secure. It proves only the necessity of some spur to ambition, such as poverty may supply. In truth, however, the capacity to be thus stimulated is also essential; and I believe that the vast majority of those who encounter early deprivations are daunted or overcome by the conflict. The fittest who survive are, of course, exceptional characters.

On the other hand, the sons of wealth must possess in themselves motives and powers sufficient to take the place of the pressure of necessity, if they are to utilize fully the advantages which wealth can give. Parental indulgence and social pleasures combine to weaken the motives for industry; and it is creditable to the sons of our rich men that so many of them turn out even respectably, and that a steadily increasing pro-

portion are acquiring the eminence which ought to be the result of their early advantages.

2. But of this number, embracing many illustrious contemporary names, comparatively few have continued in the business of their fathers. The son of a banker or merchant wins distinction in literature or learning, making for himself a new career. Or, if he engages in the paternal business, it is not likely that he will find therein the full sphere for his cultivated powers. His "vocation" must be supplemented with an "avocation," if he is to satisfy and utilize his whole nature.

3. Finally, of those who have, from the beginning, zealously prepared themselves for a special, congenial, and (so to speak) inherited career, not all, by any means, will prove adequate to its practical demands. Neither scholastic equipment nor high ambition nor good intentions nor hard work will supply the place of that genius of administration which is essential always, and must needs be the greater, the wider the field, and the more far-reaching the purposes, of its exercise.

The fulfillment of the above-named conditions does not of necessity imply credit to their subject, except so far as credit is deserved by the utilization of opportunity. Eckley B. Coxe was fortunate in many respects; and his fortune offers no model for voluntary imitation. But his example lies in the fact that he was equal to his fortune. Born to a hereditary mission, he prepared himself thoroughly for its demands; he made his conception of it wide and wise; he discharged it with honor, skill and success; and he made it the center of a fruitful, helpful life.

From the standpoint of an owner of coal-lands, he realized the pernicious effects of the system of short leases to individual operators, upon royalties determined by the amount of coal marketed, without reference to the proportion lost in mining and preparation. The subject of this waste occupied his mind continually, and he made warfare upon it, not only as a mine-owner, but as an engineer and patriot. He sought to diminish it by an adequate exposure of its amount, economic significance and causes; by better forms of tenure and customs of trade; by improved mining methods and machinery; and by more thorough education of both managers and workmen. All his activity, from the benevolence of his private life to his

public work for technical education and legislation, could be traced to his business experience, which, interpreted by his generous spirit, made him a greater benefactor than the most unselfish of impracticable theorists; saved him from attempting the impossible; and enabled him to teach by object-lessons what should, and could, be done.

In April, 1871, Mr. Coxe united with R. P. Rothwell and Martin B. Coryell in issuing the circular which stands at the beginning of the first volume of our *Transactions*, and in response to which the American Institute of Mining Engineers was organized at Wilkes-Barre, May 16, 1871. It is worthy of note that in this circular the economical mining and preparation of coal were specified among the subjects to which the attention of such a society could be profitably devoted. At this first meeting, in fact, the keynote was struck by Mr. Rothwell in a paper on "The Waste in Coal Mining," at the close of the discussion of which a committee was appointed, with Eckley B. Coxe as chairman, to consider and report on that subject.* Two months later, at the Bethlehem meeting,† Mr. Coxe presented the preliminary report of this committee, which laid out the field of its investigations in three divisions, covering respectively the waste of coal in mining, preparation, and transportation, and indicating the lines of profitable inquiry and possible reform. That this committee never presented to the Institute a complete final report does not prove the failure of its scheme. Much fruitful discussion, dating from that beginning, has appeared from time to time in the *Transactions*; and Eckley B. Coxe lavishly devoted much labor and money to the collection of facts, at home and abroad, bearing upon the great subject which, in one form or another, occupied him all the rest of his life. These accumulated materials were too voluminous for an Institute paper. Some of them found publication in the volumes of the Pennsylvania Geological Survey,‡ the active young workers of which were members of the Institute; a valuable portion was embodied in Mr. Coxe's individual papers on the preparation and utilization of coal; and the fruit

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* *Trans.*, i., 9, 55. † *Id.*, i., 12, 59.

‡ See especially Volume A 2 (1881), on "The Causes, Kinds and Amount of Waste in Mining Anthracite." At page 29 a contribution from Mr. Coxe begins.

of all appeared in the report of the State Commission, constituted in 1889, of which he was a member.* Of the original Commission, Mr. Coke was the only member who lived to see the completion of the report, Messrs. J. A. Price, of Scranton, and P. W. Sheafer, of Pottsville, his first colleagues (both members of the Institute), having died during their official labors. It is evident from this report, as from his later papers and addresses, that Mr. Coxe's study of the subject had led him to select, as the most important of all the practicable measures of economy, the utilization of the smallest sizes of coal, such as had been allowed for many years to be lost in the slaty waste. His improved machinery for preparation, described in his paper on "The Iron Breaker at Drifton," etc.,† and his improved apparatus for the combustion of small coals, described in his paper on "A Furnace with Automatic Stoker," etc.,‡ indicate the two lines of experiment in which he was ultimately absorbed; and his work in the latter direction is admirably summed up in the paper§ which he read at Providence, R. I., before the New England Cotton Manufacturers' Association, April 24, 1895, less than three weeks before his death. The possible distrust with which a consumer of coal might listen to the advice of a producer is humorously anticipated by the line from the *Æneid*, prefixed to this paper as a motto:

Timeo Danaos et dona ferentes.

But such a distrust must have been dispelled by the frankness of the opening sentences:

"It may seem curious that a person whose life has been spent in mining and marketing coal should appear before this association to discuss the economical production of steam, involving, as it does, either the use of less fuel or fuel of less value. But I am convinced that the more valuable a ton of coal becomes to our customers, the more in the end will be our profit from it."

This characteristic utterance might serve as the motto of the

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* *Report of Commission Appointed to Investigate the Waste of Coal-Mining with the View to the Utilizing of the Waste.* Philadelphia, 1893.

† *Trans.*, xix., 398.

‡ *Id.*, xxii., 581.

§ "Some Thoughts upon the Economical Production of Steam, with Reference to the Use of Cheap Fuel, By a Miner of Coal."

life of Eckley B. Coxe—a life which solved the antagonism between altruism and egoism, not by sacrificing either, but by viewing both upon the higher plane where they are one. "Enlightened selfishness," if it be only sufficiently enlightened, and command a sufficiently wide horizon, is true benevolence. "There is that scattereth, *and yet increaseth.*" The dividend of what we invest in mankind is greater than the principal of what we hoard. *This* sort of book-keeping also should be more generally understood.

Here I may appropriately return to the connection of Mr. Coxe with our Institute; for this is the principle upon which the Institute was founded by him and his associates, namely, that the mutual exchange of knowledge is better, both for the whole profession and for each individual member of it, than the attempt to secure its advantages by hoarding single fragments of it as private treasures.

Beyond doubt, if professional eminence alone had been the controlling consideration, Eckley B. Coxe would have been the first President of the Institute. But that honor was conferred upon David Thomas—"Father Thomas"—in deserved recognition of his services as a pioneer in the iron-manufacture, and in expression of the equal recognition which the Institute has always extended to men of practice, as well as men of science. Mr. Coxe became at once a Vice-President, and served as such six years. In 1878 and 1879 he was elected to the presidency; in 1884 and 1885, and again in 1889 and 1890, he was a Vice-President; so that for twelve out of the twenty-four years of the history of the Institute, he was actively engaged in its management. I remember well how his faith in it was illustrated in the early days, when the publication of the first volume of the *Transactions* was under consideration by the Council, and the question of expense was a pressing one. There was not money enough in the treasury to pay for even a small edition; the number of members, though rapidly growing, was not more than one or two hundred; and the Council would be individually responsible for any debts contracted. In justice to all, it must be said that they were unanimously ready to take the responsibility of the publication; but it was on the earnest proposal of Mr. Coxe, backed by the offer of his personal guaranty, that they finally decided to print an edition of 1500 copies.

I am happy to add that the prosperity of the Institute more than fulfilled his high anticipations.*

Mr. Coxe's contributions to the *Transactions* are enumerated in an appendix hereto. I would here call special attention to his presidential addresses, particularly that on "Secondary Technical Education,"† and also to his remarks in the joint meeting of the Society of Civil Engineers and this Institute, in 1876.‡ In both of these, as in other utterances, he emphasized an aspect of the subject different from that which others had considered. They were chiefly concerned to inquire how the graduates of technical schools could be brought into the necessary contact and acquaintance with practice. He, while not in the least out of sympathy with the study of this problem,§

* In this connection, I am reminded of a characteristic incident in June, 1876, when, in connection with the Centennial Exposition, a banquet was given by the Institute to its foreign guests, at Belmont Mansion. in Fairmount Park (*Trans.*, v., 10). The number of guests was so large, and the projected entertainment so lavish, that an attempt to defray the expense by the usual methods of a subscription dinner would have made the price of tickets prohibitory to many of the members. But into our deliberations came Eckley Coxe, bringing a breeze with him, as usual, and saying impetuously: "Look here! my dear old governor has given me a lot of money to entertain foreign swells with. Now I don't want to have the trouble of giving separate dinners and things. What I want is, that you fellows should get up the best thing you can, put the tickets for members at a price that they can all afford, draw on me for the extra expense, and say nothing about it!"

This generous offer was accepted; the banquet, arranged without regard to expense, went off most brilliantly; the Institute reaped much glory from it; Eckley Coxe paid with great delight a great bill; and, conceiving myself to be released by his death from the promise of secresy, I tell the story as an apt illustration of his character.

† *Trans.*, vii., 217.

‡ It is to be regretted that the proceedings of this memorable meeting were not published in the volume of the *Transactions* of the Institute. The meeting was a consequence of the admirable presidential address of Alexander L. Holley, on "The Inadequate Union of Engineering Science and Art" (*Trans.*, iv., 191), delivered at the Washington meeting of February, 1876, which aroused an interest transcending the boundaries of our membership. At the joint sessions of the two societies above named, devoted to the discussion of this subject, the opinions and suggestions of the highest authorities in technical education and in engineering practice were presented. The proceedings were published by the Institute in a separate pamphlet of 146 pages, entitled "Discussions on Technical Education," etc., copies of which can still be obtained at the office of the Secretary. It is not too much to say that this pamphlet constitutes one of the most important contributions ever made to the literature of technical education.

§ See the paper of Prof. H. S. Munroe, of Columbia College, on "A Summer School of Practical Mining," *Trans.*, ix, 664, which shows the important part

suggested another, of correlative and indeed co-operative importance, namely: How can the sons of practice be raised to a comprehension of the value of theory, and made to do intelligently what they are accustomed to do mechanically? Others contemplated the addition of some skill to scientific knowledge. He contemplated also the addition of some scientific knowledge to skill. To the members of this Institute, which brings together in harmonious communion the two classes whom Holley called "the school-men" and "the practicians," it need not be argued that both processes of education are alike important. The one which he thus emphasized, he illustrated also in practice, by the establishment at Drifton of a school for the education of the sons of working miners. This school, which was maintained for years under the direction of Oswald J. Heinrich, an honored member of the Institute, was succeeded, after Mr. Heinrich's death, by "The Mining and Mechanical Institute of the Anthracite Coal-Region of Pennsylvania," organized in 1893, chartered in 1894, and located at Freeland, Luzerne county, Pa. This institution comprises both night and day instruction, and possesses a faculty of six competent teachers. Besides liberal pecuniary support in other ways, and the valuable contribution of his own time and thought in frequent practical lectures to the students, Mr. Coxe established a scholarship prize of $300, to be awarded to the pupil in the night school having the best record at the close of the year. This sum is to be used (by relieving the recipient from the necessity

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taken by Mr. Coxe in this new form of technical instruction. The first summer school was held at Drifton in 1877; and Prof. Munroe quotes from the *Engineering and Mining Journal* of August 11th of that year, an account of its proceedings, including the following:

". . . . Mr. Coxe, also, during their stay, gave the students several informal talks of great practical value on the mining of anthracite, installation of machinery, management of men, necessity for scientific book-keeping, discussion and analysis of mine accounts, etc.

"The miners and the mine-bosses of the Cross Creek collieries have taken great interest in the experiment; and to their cordial and hearty co-operation in the carrying out of all the details of the plan is due in no small degree the very gratifying success obtained.

"It is, however, to the interest taken by Mr. Eckley B. Coxe in this new departure in mining education that its success is in a large measure due; for he not only encouraged the idea when proposed, but placed his collieries—probably the finest in this country—at the service of the school, and gave such instructions as secured for the students a favorable introduction to the miners."

of day-labor) to further his higher education in the day-school during the year following; and if the prize-scholar then enters the Lehigh University, or any similar institution, he will receive towards his tuition and expenses the sum of $300 per annum for the four years of his course; thus making in all the amount of $1500, bestowed upon one meritorious student. Mr. Coxe also provided for a regular annual contribution of $1200 to the general expenses of the institution.

The mention of Lehigh University leads me to remark here, that he was for many years a trustee of that institution, and ardently interested in its prosperity. He heartily rejoiced in the appointment of Prof. T. M. Drown to the presidency, believing it to be the beginning of a new era of honor and usefulness for the University. The declaration which I have elsewhere* quoted from one of his private letters, written shortly before his death, to the effect that he had now two objects to live for, "Lehigh University and the burning of small sizes of anthracite coal," is thoroughly significant of the range of his views and sympathies.

For the last year or two of his life, Mr. Coxe was one of the Commissioners of the Second Pennsylvania Geological Survey. He had taken a deep interest in that work from its beginning in 1874, under the charge of his life-long friend, Prof. Lesley, and had aided it with time, money, influence, and his own most valuable surveys in the anthracite-region. In the spring of 1893, no legislative appropriation having been made for the continuance of the survey, it seemed probable that the commission would be obliged to wind up its work and disband. But Mr. Coxe contributed the cost of keeping the office open, with one trained assistant, until the next meeting of the legislature, when he hoped that the Survey would be placed upon a permanent basis. Subsequently he made another liberal contribution towards the expense of publishing the third volume of the Final Report. Prof. Lesley's assistant, communicating these facts in a private letter, adds that, although the money given by Mr. Coxe was considerable and exceptional, his interest and influence have been of still greater value, and, should the survey be kept together, and hereafter placed upon a permanent basis, that result will have been due chiefly to him.

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* *Engineering and Mining Journal*, May 18, 1895.

The political career of Eckley B. Coxe was not as conspicuous as it would have been, had he not belonged to a party usually in the minority in his State. I must be permitted to say, that perhaps this circumstance may be considered fortunate. The work of his life was so profoundly important and beneficent that we may well doubt whether it could have been adequately replaced by anything which he could have accomplished by diverting his energies to another sphere.

A sincere believer in the necessity of government through political parties, and a loyal "party man," without any trace of the impracticable "doctrinaire," he was at the same time an earnest advocate of a reformed civil service, as opposed to the "spoils" system of distributing offices, and he stood firmly for "tariff reform," the decentralization of governmental power, and the old fashioned "Democratic" doctrines. In national politics, his influence was not insignificant, though he kept himself usually in the background. He labored for the nomination of Mr. Tilden in 1876 and 1880, and for that of Mr. Cleveland in 1884, 1888 and 1892. In 1884 he was chairman of the Pennsylvania delegation to the national convention of his party at Chicago.

When he was first elected to the Senate of Pennsylvania (the only political position for which he was ever a candidate), he startled and amused the politicians by resigning his seat, because he could not conscientiously take the oath, prescribed by the Constitution of the State, concerning expenditures directly or indirectly made to promote his election.* Nobody believed

* The oath which he declined to take denied all expenditures "except those expressly authorized by law." The Pennsylvania statute of 1874 specified certain expenditures as alone "authorized." Mr. Coxe had expended money for purposes entirely proper, but not included in this list, as he construed it; and therefore he would not take the oath. His refusal to do so was misunderstood by many, who suspected some deep underlying scheme; but this suspicion, soon dispelled by the influence of his personal character, gave place to a universal recognition of his great ability and his high standard of honor. Even members of the opposite party were often glad to support him, outside of partisan lines; and he was thus enabled to promote much good, and to check much bad legislation. I take the liberty of quoting, in this connection, from the private letters of one who knew him intimately, a passage which seems to me felicitously true:

"Several persons of education and good position have said to me that they felt that their views of life had been changed since they knew him, and that their standard of honor and honesty had been raised by their association with him. I have heard the term, 'moral upas-tree' applied to a person whose influence on

for a moment that he had violated the spirit of the legal prohibition; and his scrupulous sense of honor as to the letter, coupled with his naive apology for neglect to notice it earlier, while it provoked a smile, served also to increase the general esteem with which he was personally regarded. He was, of course, triumphantly re-elected, and served his term as Senator. As a member of the minority, he could not be expected to take an influential part in shaping legislation; but on non-partisan questions, his character and experience had their weight. I notice particularly his remarks in the discussion of the "Voluntary Trade Tribunal Statute," enacted in 1883. While frankly assuming the standpoint of a large employer of labor, he showed a sympathetic appreciation of the feelings of the workingmen which could not have been surpassed by one of themselves; and, while cordially supporting the bill in question, he evidently did not share the sanguine expectations of those who expected it to prove a complete settlement of the "labor question." His sane and reasonable view is indicated in the following extract from one of his speeches:

"Though not pretending to be a workingman, or in any way his representative, but, on the contrary, a large employer of labor of all kinds, I feel and admit that he has equal rights with me. What he properly demands, and what he will have, is justice. To be satisfied, he must feel that the bargain is fair, and that it has been reached in an honorable way, without any resort to force or coercion. He cares more for this than for a slight addition to or a deduction from his daily pay. Where the workingman does not get his just dues, trouble must ensue, and capital must pay its share of the bill, which is often a large one.

"I do not claim that this bill is perfect; I do not claim even that it is the proper remedy; yet I do say that it is a step in the right direction. There is no royal road to the settlement of the great questions of capital and labor. It is like cutting a road through a thick forest. We must push ahead, and as, from time to time, a ray of sunlight breaks in and shows that we are travelling in the wrong direction, we must either retrace our steps or turn off to what seems to us the proper point. That the bill, if enacted into a law, will be a complete success, no man can prophesy. We hope for the best, and we can correct our errors as we advance. . . . . The greater the security and the prosperity and justice that a government affords to the working-classes, the greater will be their feeling of devotion to it, and the less will be the danger of communism and socialism and all its attendant evils; and I feel that while this bill may not produce as great results as some of us hope, yet it will be a forerunner of something which, in the distant future, may settle forever the main dispute that now divides the people of this country."

The sincerity of these sentiments was realized by the work-

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others was bad. Eckley B. Coxe was a moral Eucalyptus tree, having the power to purify the moral atmosphere, and to make those who approached it better."

ingmen in his employ. But that did not prevent him from suffering, like other employers, the evils and losses due to strikes. Under the modern form of labor-organization, sympathetic strikes may be ordered against the best of employers, and without any local grievance. Mr. Coxe recognized and denounced the injustice of this system; but he took it good-naturedly, and did not allow it to embitter his personal relations with his employees. When a long strike reduced their families to want, his house was the center of a generous relief. He could not fight women and children.

Even violence he would not meet with violence. He was the intimate friend and strong supporter of Franklin B. Gowen in the glorious struggle with the "Molly Maguires," and the following passage from his eulogy of Mr. Gowen (*Trans.*, xviii., 618) may be taken as a revelation of his own character:

"But Mr. Gowen's greatest claim to the admiration and gratitude of his countrymen was due to the courage, perseverance and skill with which he pursued and destroyed the organization of assassins known as the Molly Maguires, terminated the reign of terror they had maintained in the anthracite regions, and inaugurated, in its place, an era of peace, safety and order, which has not yet passed away. The full measure of his merit in that undertaking cannot be appreciated without considering that he vindicated law as well as justice. The condition of affairs had become so intolerable that many citizens of the better class were seriously considering the formation of a 'vigilance committee,' to take the law into its own hands, and protect by violence the life and property which violence had put in daily peril. Without stopping to consider in what case this extreme resort might be justified, we must admit that it is an evil, even when it is a necessary one, and that, after the achievement of the main ends sought through such measures, there must follow a period during which the confidence of the citizen in the regular forms of law is weakened, to be restored by slow degrees only. Mr. Gowen, when consulted in the matter, invariably and emphatically refused to take part in unlawful means of any kind, and maintained that the remedy should be, and could be, found in the regular proceedings of the established courts of justice. To the demonstration of this proposition he devoted his great abilities with a splendid audacity, pertinacity and acuteness, risking his life freely, and knowing neither discouragement nor rest until he had brought to trial, conviction and execution the chiefs and agents of the conspiracy. The moral effect of his campaign was worth that of a hundred vigilance committees. It is gratefully recognized to-day in thousands of peaceful homes throughout the coal-mining districts of Eastern Pennsylvania, and, indeed, I should scarcely be guilty of exaggeration if I declared that the example set by Franklin B. Gowen, in the suppression of anarchy and crime by the ordinary methods of civilized government, has been an example to the friends of order and justice throughout the world."

Himself an earnest Democrat in politics and a Protestant Episcopalian in religion, he never permitted the politics or reli-

gion of any man to be interfered with or to influence his treatment in matters of business or discipline.

The company which he directed has maintained for many years an accident fund, to which nothing is contributed by the men, the company supplying all the money, and retaining the entire control. In case of death or disability resulting from accident (not from illness) during actual work for the company, the following amounts are paid: *In case of death*, $50 allowance for funeral expenses; $3 per week for one year to the widow (unless re-married sooner); and $1 per week to each child below twelve years of age, until that age is reached. *In case of disability*, $5 per week to a man, and $2.50 per week to a boy, until able to do light work.

In 1883 a hospital was built at Drifton by the company, and thereafter supported by the company and by members of the Coxe family. It could accommodate 35 patients, and was used exclusively for the employees of Coxe Brothers & Co., injured by accident. When the State hospital for miners was opened at Hazleton, about 1891, the Drifton hospital was closed.

In 1879 the company and the family* erected at Drifton a large building, containing on the upper floor a hall for meetings, lectures and entertainments (including Christmas festivals, at which all the children of employees were personally remembered in gifts from the Coxe family), and rooms on the lower floor for the technical school already mentioned. This hall was destroyed by fire in 1888, a year or two after Mr. Heinrich's death; and the school, which had been continued under the charge of Mr. John R. Wagner, was discontinued shortly after, to be replaced by the Institute at Freeland, near by, of the

* It is impossible to separate, in such matters, the part taken by the family as individuals from that of the company which was their business representative, or the part of Eckley B. Coxe himself from that of the kindred who so heartily united with him in every good work. While I comply with their own desire, as well as with the general rule of justice, in ascribing to him the credit for the undertakings of all kinds in which he was, so to speak, the official leader, I cannot forbear to say here, once for all, that I do not believe he could have accomplished, and I scarcely believe he would have undertaken, so much without the cordial and effective support of his wife and his brothers and sisters. This qualification does not in the least detract from his fame; and, on the other hand, it furnishes the assurance that his wisely benevolent schemes and policies will not end with his death.

night school of which Mr. Wagner is still principal, in addition to his professional duties as an engineer of the company.

A library and reading-room, provided with current newspapers and periodicals, and open every evening, as well as for some hours on Sundays, has been maintained at Drifton for many years by the Coxe family.

I have taken the pains to state at some length these features of the dealings of Eckley B. Coxe with his employees, because they illustrate his peculiar practical wisdom. This is not the place to discuss at length the great question with which they are connected, as to the ultimate relations between capital and labor. Nor would it be fair to set up this example as one to be imitated in all cases—especially by corporations, which can scarcely expect to establish personal relations of confidence (beyond the element of justice and fair play) with their employees. What I wish to point out is:

1. Eckley B. Coxe did not believe that the general problem of labor had been solved, if, indeed, it ever could be solved, by a general formula. As his public utterances abundantly showed, his own view was, that no solution impairing the right of free and responsible contract would be permanent or beneficial.

2. But he did not conceive that the moral obligation of human fellowship was suspended, either by legislation or by the lack of it; and he felt himself bound to discharge this obligation, whatever might be the existent legal organization of society.

3. On the other hand, he took pains to keep business (*i.e.* justice) separate from friendship (*i.e.* charity). It is the great difficulty of all organized attempts for the "benefit" of workingmen, that the workingmen themselves are prone, justly or unjustly, to distrust them. As a miner once said frankly to me, "We 'union men' don't like these building and loan associations, and benefit funds and what not. We think they are devices to make the workingmen contented when they ought to be discontented; to make them unwilling and unready to strike, when they ought to be willing and ready; to cajole them into accepting lower wages than they are entitled to receive." If he had known Latin, he might have added, *Timeo Danaos et dona ferentes!*

That this distrust is unreasonable, does not alter the fact that

it exists, and must be dealt with. And, while it lasts, the administration of "benefit" schemes of all kinds, to which both employer and employees contribute, and in the management of which the latter share, will continue to be difficult. Personally, I am by no means convinced that such joint operations are going to prove themselves the fittest, and therefore to survive. Possibly workingmen, like other citizens, had better be left to secure their own insurance, etc., through agencies in which their employers, as such, take no part. At all events, in the case now before us, Eckley B. Coxe adopted the policy which best suited his position. Dealing with his employees in business matters as equals, he kept wholly within his own control whatever was done for their benefit from motives of friendship. But he proved to them continually, in innumerable ways, that the friendship was real and sincere.

4. Apart from the relief of actual (and undeserved) need, he confined his benefactions chiefly to the gifts of knowledge and of opportunity—the only donations which do not pauperize the recipient.

But no analytical catalogue can explain the effects in which a noble, ardent, sympathetic, Christian character was the most potent factor. To imitate Eckley B. Coxe, one must not merely copy what he did, but be what he was. How many of us have experienced his cordial brotherly sympathy and aid! To how many of us (as to me) the very sight of his shining face was a joy and inspiration! The Germans have a word, *lebensfroh*, "glad to be alive," for which I can think of no English equivalent, but which his countenance seemed always to utter. It was as if he said aloud, when he met a friend, "Is not life worth living? Look at me!"

To this radiance of personal manner, was added a peculiar force of speech. His conversation or extempore oratory was *sui generis*—not finished according to the rules of art, but impetuous and fragmentary, abandoning one sentence to begin another, yet full of ideas, and never failing to convey them effectively. He was the delight of audiences and the despair of reporters. His off-hand contributions to oral discussion sparkled with epigrammatic wisdom, the characteristic form of which was unfortunately, as a rule, not preserved when he subsequently revised his remarks for publication. Now and

then, a specimen remains; and I take at random the following examples, one or two of which I quote from memory:

"The problem (of engineering education) seems to be: *Not knowing exactly what you want to do or the material you have to do it with, what is the best way of doing it?*

"This work (a long steam-pipe across country) was not put up in New York City, where people do not want their spectacles covered with moisture. It is put up in the woods. It is peripatetic engineering. If you made a plan for it, before you had your plan completed your practical man would have it up and in running order."

"When you go out into practical life, do not expect to get the place of Chief Engineer of the New York Central and Hudson River Railroad—you will have to wait awhile. If you did get it, it would be the worst thing for you; you would either go crazy, or be discharged within a month. A doctor, on graduating, goes to a hospital, where he works for nothing, and does no harm—or, if he does harm, no one knows it. The same thing is true, in one sense, of the engineer. I would rather have a young man get a place where he has opportunities, and not too responsible a position—one where he would not be overburdened."

"Young engineers should not put on airs. When you are exhibiting your plan for the works, don't pretend that you invented the fire-place or the chimney."

"The great difficulty that an employer has in managing men is to manage himself. He is apt to think that he knows more than all his men: but though a man be educated, wise and careful, some one on the other side will catch him if he trips."

"Eternal vigilance is said to be the price of liberty: it is also the price of cheap steam."

"The usual practice in the preparation of anthracite is to take out as much of the slate as practicable, and persuade the consumer to accept the rest as coal.

He died May 13th, of pneumonia, after a very brief illness; and the interval before his funeral on the 16th was too short to permit any formal participation on that occasion by all the numerous organizations of which he was an honored member, and which he had laid under obligations of gratitude. But the multitude of mourners comprised many leading representatives of such organizations; and I am glad to say that the delegation from this Institute, headed by his life-long friend, John Fritz, was worthy in numbers and in character of the love and

sorrow expressed by its presence. Yet, after all, it was the man, not the member of any society, who was missed and mourned that day; and all sense of official relations was lost in the consciousness of personal grief.

According to his own request, Mr. Coxe was buried in the midst of those for whom he had labored; in front of the church which he had built; close by the Sunday-School where his wife continues with three hundred children her labor of love; and but a stone's throw from the office which was the center of his beneficent activity.

On May 30th, "Decoration Day," the "Grand Army" veterans, besides honoring the graves of those who had died for their country, held a special service at the grave of Eckley B. Coxe, rightly deeming that they also are worthy of grateful remembrance who have given their lives in love and labor for their fellow-men. I venture to introduce here, as a fitting conclusion to this sketch of his life, the lines which were read on that occasion, expressing, as they do, my conception of his highest claim to praise.

AT THE GRAVE OF ECKLEY B. COXE.

DECORATION DAY, 1895.

O near and dear to many a grateful friend,
Yet nearest, dearest, noblest held by those
To whom thou didst a brother's hand extend,
A brother's heart disclose!—

Thy comrades, in whatever lowlier guise,
Who saw thee not in pride of state above,
But, face to face, caught from thy shining eyes
The level glance of love!

True knight of love, in stainless armor bright
Full-clad, and ardent with all wrong to cope,
And wearing ever in the front of fight
The crested helm of hope!

True steward of the trust of earthly power,
Nor weak to waste, nor miserly to save,
But wisely using all, until the hour
When He should ask, who gave!

Strong to pursue the path by Science trod;
Strong to achieve what lies in human ken;
Yet strongest by thy steadfast faith in God
And in thy fellow-men!

Among us still thou wouldest fain abide:
Witness, in proof, thy grave among us made—
Albeit we know thy spirit glorified
Not in the grave was laid.

Yet precious evermore shall be the spot
That hides the earthly form to us so dear;
A sign, by children's children unforgot
That still thy soul is near;

An inspiration, stirring sons of earth
To do for each and all what brothers can;
A memory and a presence, holding forth
The pattern of a Man!

A thousand voices over land and sea
Acclaim thy praise, and make their sorrow known;
Yet not less welcome to thy heart will be
This tribute from thine own!

The impetus and momentum of such a life are not to be arrested by the physical accident of death. As was graphically said on a similar occasion, "an avalanche that has slid a mile will not be stopped by a gravestone." Our greatest debt to such strong, bright spirits as Eckley B. Coxe is the conviction which they produce in our souls of the victorious persistence of life, and thus of the truth of the Life Eternal.

Appendix I.

Publications.

The translation of Weisbach's *Mechanics*, published in 1870, of which the full title-page is given in a preceding page, was, I believe, the only formal book which Mr. Coxe ever produced. Of his numerous addresses and professional papers, I have prepared the following list, which is, no doubt, incomplete; and I need hardly say that corrections of it and additions to it will be gratefully received.

Date.	Title and Reference.
Oct. 25, 1870.	Mining Legislation. A paper read before the American Social Science Association.
Aug. 15, 1871.	Preliminary Report of the Committee upon the Waste of Anthracite Coal (*Trans.*, i., 59).
Feb. 20, 1872.	Remarks on Systems of Coal-Mining (*Trans.*, i., 182).
May 21, 1872.	A New Method of Sinking Shafts (*Trans.*, i., 261).
Feb. 18, 1873.	The Use of the Plummet-Lamp in Underground Surveying (*Trans.*, i., 378).
Feb. 24, 1874.	Improved Method of Measuring in Mine-Surveys (*Trans.*, ii., 219).

Feb. 24, 1874.	Remarks on Poisoning by Carbonic Oxide (*Trans.*, ii., 197).
Feb. 24, 1874.	Remarks on the Diamond Drill for Deep Boring (*Trans.*, ii, 259).
May 27, 1874.	Improved Form of Plummet-Lamp, for Surveying in Mines where Fire-Damp may be Met with (*Trans.*, iii., 39).
Oct. 27, 1874.	Remarks on Blast-Furnace Fuel (*Trans.*, iii., 183).
Feb. 23, 1875.	Remarks on the Carbonite, or so-called Natural Coke of Virginia (*Trans.*, iii., 458).
Feb. 22, 1876.	Remarks on What Steel Is (*Trans.*, iv., 337).
June 20, 1876.	Remarks on Technical Education at the Joint Meeting of the American Society of Civil Engineers and the American Institute of Mining Engineers (reported in "Discussions on Technical Education," etc.; published by the Institute, 1876, p. 88).
June 23, 1876.	Remarks on the Nomenclature of Iron (*Trans.*, v., 314).
Oct. 24, 1876.	Remarks on Anthracite-Mining in Schuylkill county, Pa. (*Trans.*, v., 416 *et seq.*).
June 19, 1878.	Engineering as a Profession. Address before the Alumni Association of Lehigh University.
Oct. 15, 1878.	Presidential Address on Mining Engineering as a Profession (*Trans.*, vii., 103).
Oct. 15, 1878.	Note upon a Peculiar Variety of Anthracite (*Trans.*, vii., 213).
Feb. 18, 1879.	Presidential Address on Secondary Technical Education (*Trans.*, vii., 217).
Sept. 16, 1879.	Presidential Address on the Object of the American Institute of Mining Engineers (*Trans.*, viii., 126).
Feb. 17, 1880.	Note on the Use of Carbonate of Soda for the Prevention of Boiler-Scale (*Trans.*, viii., 279).
Feb. 15, 1881.	Letter on Summer Schools of Practical Mining (*Trans.*, ix., 664).
Oct. 11, 1883.	Memorial Address on Founder's Day, Lehigh University.
1883.	Remarks in the Senate of Pennsylvania on Arbitration.
Feb. 13, 1885.	Some Thoughts on the Practical Management of Public Works. Lecture before the Engineering Society of the School of Mines (*School of Mines Quarterly*, vi., 251).
Nov., 1886.	The Tendencies of Modern Engineering. Lecture at Sibley College, Cornell University (*Scientific American*, Nov. 13, 1886).
Aug. 10, 1887.	Engineering. Vice-President's Address in Section D. Am. Asso. Advancement of Science (*Proc.*, vol. xxxvi., p. 147).
Feb. 18, 1890.	Biographical Notice of Franklin B. Gowen (*Trans.*, xviii., 618).
Sept. 29, 1890.	Remarks on the Electric Transmission of Power (*Trans.*, xix., 286).
Sept. 29, 1890.	The Iron Breaker at Drifton, with a Description of Some of the Machinery Used for Handling and Preparing Coal at the Cross Creek Collieries (*Trans.*, xix., 398).
Oct. 6, 1891.	Remarks on the Preparation of Small Sizes of Anthracite (*Trans.*, xx., 613, 619).
Oct. 6, 1891.	Remarks on Centrifugal Ventilators (*Trans.*, xx., 670).
Oct. 6, 1891.	Remarks on Tests of Structural Wrought-Iron and Steel (*Trans.*, xx., 710).
Oct. 23, 1891.	Economy of Steam in Engines and Generators. Lecture to the students of Sibley College, Cornell University (reported in *The Crank, the Sibley Journal of Engineering*, vol. vi., No. 2).

May, 1893.	Report of the Pennsylvania State Commission on the Waste of Coal-Mining.
Aug., 1893.	A Furnace with Automatic Stoker, Travelling Grate and Variable Blast, Intended Especially for Burning Small Anthracite Coals (*Trans.*, xxii., 581).
Aug., 1893.	Remarks on Fire-Damp in Mines (*Trans.*, xxii., 729).
Dec. 4, 1893.	The Use of Small Sizes of Anthracite Coal for Generating Steam. President's Address before the American Society of Mechanical Engineers (*Trans. Am. Soc. Mech. Eng.*, vol. xv., 36).
Dec. 4, 1893.	Remarks on the Conveyance of Steam in Pipes (*Id.*, vol. xv., p. 547).
Dec. 4, 1893.	Remarks on the Protection of Plungers against Acid Waters (*Id.*, vol. xv., p. 599).
1893.	The Regulation of Wages. Lecture before the Franklin Institute, Philadelphia.
1894.	Lectures at the Mining and Mechanical Institute of the Anthracite Coal Region.
June 5, 1894.	Technical Education. President's Address before the American Society of Mechanical Engineers (*Trans. Am. Soc. Mech. Eng.*, vol. xv., p. 655).
Jan., 1895.	Modern Education. Lecture at Lehigh University.
April, 1895.	Some Thoughts upon the Economical Production of Steam, with Special Reference to the Use of Cheap Fuel, by a Miner of Coal. Paper read before the New England Cotton Manufacturers' Association, at Providence, R. I. (condensed in the *American Machinist* for May 23 and 30, 1895. The number for May 23 contains also an obituary notice of Mr. Coxe).

APPENDIX II.

INVENTIONS AND EXPERIMENTS.

In the minute adopted by the Faculty of Lehigh University occurs the following just and felicitous passage:

"The death of Mr. Coxe closes a period of private experimental engineering that has no parallel in any country. Of ample fortune, and with unlimited abilities for research, he devoted large sums to the solution of problems of public utility, and freely gave the results to the world."

In these labors he surrounded himself with skillful assistants, and with the best scientific and literary apparatus, including a chemical and physical laboratory and a scientific library unsurpassed in its kind, which comprised full sets of the publications of the principal mining and engineering societies of the world, and of the chief scientific journals, as well as many rare books, and the whole of the scientific library left by Prof. Julius Weisbach, of Freiberg.

Among the inventions of Mr. Coxe were the following:

Long steel tapes, instead of chains, for mine-surveying; compensating-rings for the plummet-lamp; improved plummet, with safety-lamp attachment; improvement on Leslie's micrometer; Coxe's micrometer; automatic slate-picking chute; corrugated rolls for breaking coal; improved coal-jigs; gyrating screens; improvements in pumps; application of the gyrating-screen motion to the drilling

of metal in the machine-shop; grease packing for plunger-pumps; the use of a solution of chloride of zinc for the mechanical separation of coal and slate in laboratory-tests; and the mechanical stoker, which embodies many separate inventions.

Most of the above are described in our *Transactions* or in those of the American Society of Mechanical Engineers. Many of them were perfected only after long series of experiments, and the number of experimental investigations which bore no such perfected fruit was, of course, larger still. He had been for years studying in this way the combustion of coal in furnaces attached to boilers, and in a few months more would have arrived at definite conclusions which he could publish. But the "mechanical stoker" is the only complete result of his researches in this department. Another subject which had occupied him in recent years was the application of compressed air to machinery in the mines.

The records of the U. S Patent Office show 111 patents issued to Mr. Coxe either directly or as assignee of the assistants who worked under his instructions. Most of them are dated in and after 1890. The details and use of the mechanical stoker are covered by 73 patents; the gyrating screen has 10; the corrugated breaker-rolls, 3; and the remainder refer to various features of colliery- or breaker-machinery.

That Eckley B. Coxe protected by patents a portion of his inventions does not detract in the least from the generosity and public spirit of his labors Siemens has well said that if a valuable discovery lay in the street, it would be better for the public to give it to somebody for a period than to leave it to everybody. This is especially true of improvements involving expensive machinery. Nobody can afford to be the first to make and use them, if not in some way compensated for the plant of manufacture and the cost of initial trials and failures. To "dedicate to the public" a complicated mechanical system is, in such cases, simply to throw away the labor and thought expended upon it. In the instance before us, Eckley B. Coxe, having perfected new apparatus for the preparation of coal and for the combustion of small coal, gave it to the public in the only effective way, by establishing an agency which would make, guarantee and install the necessary machinery, and teach its proper operation. It has thus become to every colliery-owner not a pretty scheme on paper, which he is kindly permitted to work out for himself, but a tangible thing, which he may inspect, acquire and use. The public thus gets what it wants, and philanthropy ought to be as well pleased as is common-sense.

Appendix III.

Official Positions and Relations.

At the time of his death, Mr. Coxe was a member of the following societies. The figures in parenthesis indicate the year of his election to membership:

The American Institute of Mining Engineers (one of its founders in 1871; Vice-President 1872, 1873, 1874, 1876, 1877, 1884, 1885, 1888 and 1889; President 1878 and 1879).

The American Society of Mechanical Engineers (member 1880, Vice-President, 1880, 1881; President, 1892 to 1894).

The American Society of Civil Engineers (1877).

The Engineers' Club of Philadelphia (1884).

The American Philosophical Society (1878).

The Franklin Institute, Philadelphia (1875).

The Academy of Natural Sciences, Philadelphia.

The American Chemical Society (1894).

The American Association for the Advancement of Science (Fellow since 1874, and Vice-President of Section D, "Mechanical Science," in 1887).

The North of England Institute of Mining and Mechanical Engineers (Life-Member, 1873).

Der Verein Deutscher Ingenieure (1893).

The South Wales Institute of Engineers (1891).

The Chesterfield and Midland Counties Institution of Engineers (Life-Member, 1877).

The Chesterfield and Derbyshire Institute of Mining Engineers (Life-Member, 1877).

The Society of Chemical Industry, London (1891).

The University Archæological Association, University of Pennsylvania (1891).

La Société Géologique de France (Life-Member, 1861).

The Society for the Promotion of Engineering Education (from its foundation in 1893).

The American Academy of Political and Social Science.

The Iron and Steel Institute, Great Britain (1890).

Le Congrés International des Accidents du Travail (1891).

Le Congrés Géologique International (1891).

The American Metric Bureau (1878).

The American Metrological Society of New York (1879).

The American Social Science Association (1875).

The American Society for the Extension of University Teaching (1891).

The American Folk-lore Society (1889).

The Historical Society of Pennsylvania 1875).

The Wyoming Historical and Geological Society (Life-Member) (1874).

The Philadelphia Society for the Extension of University Teaching (1890).

The Philadelphia Association for the Promotion of Social Science (1877).

The Philadelphia Tariff Reform Club (1892).

The Pennsylvania Museum and School of Industrial Art (1879).

Also, the following Societies, Clubs, etc.: Delta Phi Fraternity, University of Pennsylvania Chapter; the Delta Phi Club of New York; the Alumni Association of the Delta Phi, Philadelphia; the Phi Beta Kappa Society; the Rittenhouse, University, Clover, Mercantile, and Germantown Cricket Clubs, Philadelphia; the Manhattan, Engineers', and Lawyers' Clubs, New York; the Westmoreland Club, Wilkes-Barre, and a number of political associations.

In addition to those above named, Mr. Coxe held the following positions:

Trustee of Lehigh University from the beginning (Chairman of the Committee on College Administration, and member of the Library Committee).

Member of the Board of Trustees of the Pennsylvania Institution for the Deaf and Dumb (Life-Member from 1886).

Vice-President of the Mining Congress held at the Paris Exposition in 1889.

Member of the State Commission on the "Waste of Anthracite Coal" (appointed February 19, 1890).

Member of the Commission of the Second Geological Survey of Pennsylvania (1893).

President of the Board of Trustees of the State Hospital for the Middle Anthracite Coal Fields (from the beginning, December 17, 1890).

Member of the Board of Directors of the Philadelphia and Reading Railroad Company.

State Senator from the Luzerne and Lackawanna District of Pennsylvania for the term of four years from November, 1880.

A Section of Rich Patch Mountain at Iron Gate, Va.

BY E. J. SCHMITZ, NEW YORK CITY.

(Atlanta Meeting, October, 1895).

THIS section was obtained last year during an examination of the iron-ore resources of the Rich Patch Mountain region and along Craig's creek valley. The geological members below the Oriskany and Lower Helderberg are exposed along the gap in a beautiful arch, while the higher formations appear in a more or less disturbed state on both sides of this arch, and are partly covered by *débris*. I have thought it of interest to the Institute to reproduce the drawing of this section at the gap which I made in my note-book at the time of my examination; those parts of the section, which lie at both sides of the arch, are only approximated.

I give also a number of notes and measurements, referring to the Clinton formation and especially to the characteristics of the fossiliferous ore-seam opened in the various tunnels along the arch (see section).

I. *In Tunnel C:*

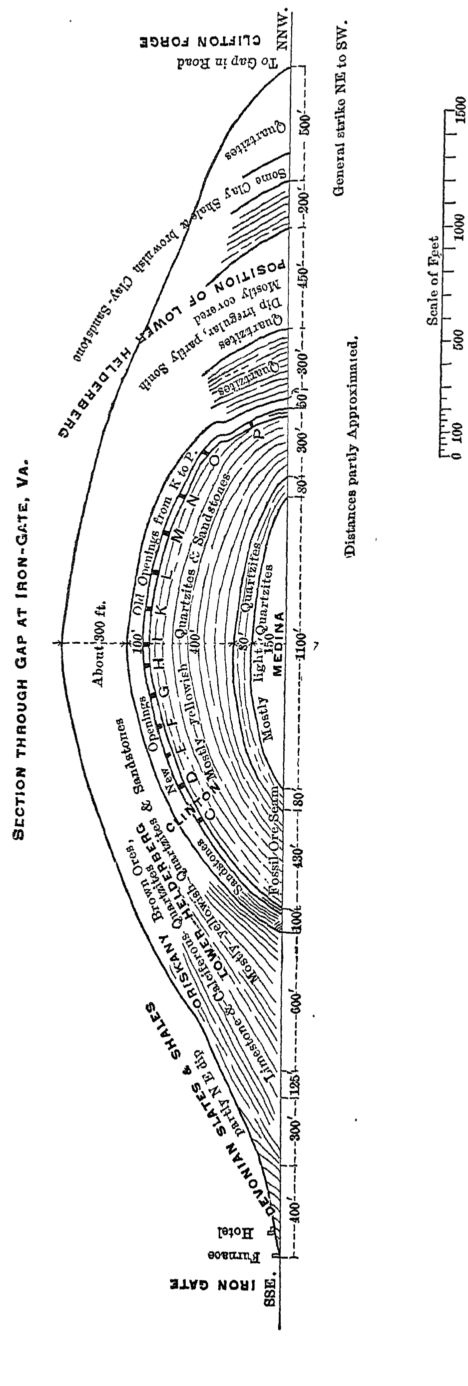
For the first 110 to 120 feet to second shute, we find the ore only here and there in small deposits, and interchanging with, or changing into, either a very siliceous limestone or a sandstone.

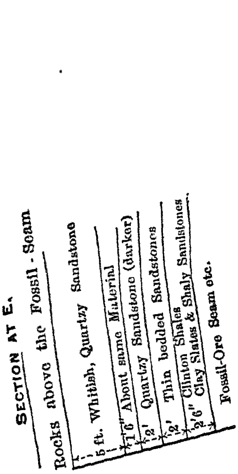
About 10 feet beyond shute we have:

Sandstone roof.
Clay-slate, 21½ inches
Brown lean ore, 1 inch.
Sandy clay, 3 to 3½ inches.
1. Dark brown ore, 3 inches.
2. Fossil-ore, changing into sandstone or very siliceous limestone, 2 inches.
3. Brown lean ore, 6 inches.
Clay and slate below, 1 foot.

194 feet from entrance:

Sandstone roof.
Clay-slate, 4 inches.
Clay 19 inches.
Fossil-ore, 6¾ to 7 inches.
Clay and ore, 1½ inches.
Lean brown ore, 2 inches.
Clay to bottom, 2 feet.


SECTION THROUGH GAP AT IRON-GATE, VA.
CLIFTON FORGE
To Gap in Road
NNW.
Quartzites
Some Clay Shales & brownish Clay-Sandstone
POSITION OF LOWER HELDERBERG
Dip irregular, partly South
Mostly covered
Quartzites
About 300 ft.
Old Openings from K to P.
New Openings
Quartzites & Sandstones
Mostly yellowish Quartzites
Mostly light Quartzites
MEDINA
CLINTON
Fossil Ore Seam
ORISKANY Brown Ores, Quartzites & Sandstones
LOWER HELDERBERG Quartzites
Calciferous Sandstones
Mostly yellowish
Limestone & Calciferous
DEVONIAN SLATES & SHALES partly N E dip
Hotel
Furnace
IRON GATE
SSE.
General strike NE to SW.
Distances partly Approximated.
Scale of Feet
0
100
500
1000
1500


SECTION AT E.
Rocks above the Fossil - Seam
5 ft. Whitish, Quartzy Sandstone
1'6" About same Material
2' Quartzy Sandstone (darker)
2' Thin bedded Sandstones
2'6" Clinton Shales Clay Slates & Shaly Sandstones
Fossil-Ore Seam etc.

II. *In Tunnel D:*

About 120 feet from entrance, near second shute:

Sandstone roof.

Sandstone, about 1 foot.

Lean fossiliferous ore, 3 inches.

Clay slate to floor, 2 feet.

From entrance of D to the above-stated section at 120 feet, I was unable in most places to find the fossil-ore. At one point I found about 3 inches of brown ore in the horizon of the fossil-ore.

III.—*In Tunnel E:*

150 feet from entrance: Sandstone roof.

Clay slate, etc., about 4 feet.

Fossil-ore, with about 1 inch of brown ore on top, $7\frac{1}{2}$ to 8 inches.

Clay and clay slate below.

200 feet from entrance is the fossil-ore, 7 inches thick.

IV.—*In Tunnel F:*

36 feet from entrance and for 10 feet along seam, we have:

12 inches fossil-ore, good, with irregular parting of 1 inch thick.

1 to 3 inches brown, shaly ore below, about 30 per cent.

The fossil-ore shows a parting of 1 inch about 3 to 4 inches from bottom, which is not persistent.

226 feet from entrance: Sandstone roof.

Clay slate and clay, 1 foot.

Fossil-ore, broken, 4 to 8 inches.

Sandy fire-clay, 6 to 12 inches.

Sandstone and shaly ore mixed with clay, 10 to 12 inches.

Soft sandy clay over 2 feet.

Average of fossil and brown ore, 11 to 12 inches.

275 feet from entrance we had fossil-ore (good), 4 inches to the left of tunnel.

276 feet to the right: Fossil-ore, 10 inches.

Clay, 4 inches.

Brown, shaly ore, 3 inches.

Mixed ore and clay, 3 to 6 inches.

384 feet from entrance, at end of tunnel:

Sandstone and clay, 4 inches.

Sandstone, surface colored by oxide of iron, 4 inches.

Clay, 11 inches.

Brown ore and clay, 15 inches.

V.—*In Tunnel G:*

94 feet from entrance, seam near bottom of tunnel (rolling):

Roof is sandstone and clay.

Ore, lean, $1\frac{1}{2}$ inches.

Fossil-ore (good), 5 inches.

Harder, lean, brown ore, 2 inches.

Clay, below.

150 feet from entrance near end of tunnel:

Roof, sandstone and clay, 3 feet.

Sandstone, colored by oxide of iron, $3\frac{1}{2}$ inches.

Clay, slate and shale, 6 inches.

Ore, partly soft and fine, 10 inches.

Fossil-ore and some brown ore below.

Clay below.

VI.—*In Tunnel H:*

About 360 feet from entrance at end of tunnel:

Roof, sandstone.

Slate (clay-slate), 1 foot.

Clay mixed with small strata of brown ore, 9 inches.

Fossil-ore, with sandstone parting, 5 to 7 inches.

Clay and mixed material to floor, 21 inches.

VII.—*In Tunnel I:*

370 feet from entrance at end of I, about 100 feet from shute to H:

Roof, sandstone.

Hard-clay slate, 2 feet.

Fossil-ore, very hard, 9 inches.

To floor, hard siliceous limestone, 1 foot, 9 inches.

75 feet from entrance of I, fossil-ore, 8½ inches.

15 feet from entrance into H:

Sandstone roof.

Shale, 8 inches.

Fossiliferous ore, hard, 6 to 7 inches.

Clay parting, 1 inch.

Hard ore, on line between fossil and brown ore, 4 to 5 inches.

Clay and sandstone, 2 feet.

At connection with H:

Sandstone roof.

Fossil-ore, 6 inches.

Mixed material, sandstone and ore, 3 inches.

At shute to H:

Fossil ore, 6½ inches.

Clay parting, 1 inch.

Fossil-ore, 5¼ inches.

Only a few of the old tunnels on the north side of the arch could be examined, as most of them were filled in. The following are my notes on these older openings:

In Tunnel K:

About 130 feet from entrance:

Sandstone roof.

Brown slates and sandstones interbedded below, 2 feet 8 inches.

Hard fossil-ore, 2½ to 6 inches.

Sandy clay, 3 feet.

At entrance: Brownish ore, 3½ to 4 inches.

Hard fossil-ore, 10 to 11½ inches.

Soft and hard fossil-ore, 10 to 12 inches.

Tunnel N.—Could be entered for about 100 feet. I found no fossil-ore, but observed some dark slaty lean ore, in about the same position as the fossil-seam.

Tunnel P.—This tunnel is claimed to be 1800 feet long. Fossil-ore is said to have given out entirely.

20 feet from entrance observed:

Sandstone roof, shale and slate below.

Brownish, slaty and sandy ore, 5 inches.

Clay parting, ½ to 1 inch.
Brown limonitic ore, 3 inches.
Fossiliferous ore, 6 inches.
Clay sandstone, 6 to 8 inches.
Banded brown ore, mixed with sandstone, 8 inches.
Clay, 3 inches.
Dark brown ore mixed with clay, 12 inches.

35 to 40 feet from entrance:

Roof, sandstone and shale.
Brown, lean, shaly ore, 3 inches.
Sandstone, colored by oxide of iron, 6 inches.
Brown shale, 3 inches.
White clay, 1 inch.
Dark, shaly, lean ore, 3 inches.
Fossiliferous ore, 8 inches.
Clay with small seam of shaly, lean ore in lower 6 inches, 13 inches.
Clay, 4 to 6 inches.
Black, slaty material, 12 inches.

Chrome in the Southern Appalachian Region.

BY WILLIAM GLENN, BALTIMORE, MD.

(Atlanta Meeting, October, 1895.)

IN their account of chromium, Roscoe and Schorlemmer (*Treatise on Chemistry*, London, 1879) state that "In 1762 Lehmann, in a letter to Buffon, *de nova minerœ plumbi specie crystalline rubra*, described a new mineral from Siberia, now termed crocoisite." In 1786 Lehmann analyzed it (Nicholson's *Jour.*, Dec., 1798) and found that it contained chiefly lead, and that the "mineralizers" were arsenic and sulphur. It was next examined in 1789 by Macquart and Vauquelin, who stated that it contained an intimate mixture of iron, alumina, and lead peroxide. The account of their work was published over the name of Macquart.

Vauquelin (Louis-Nicolas) was a Norman peasant, who had studied in the village school, and at the age of fourteen was a laboratory-boy in a pharmacy at Rouen. Later, he found his way to Paris, and in 1789 was assistant to Fourcroy, a pharmacist and a teacher of note. It was here that he met Macquart, with whom he did the work on Siberian red lead, and at this time his friend Cheradame taught him his Latin.

In the year 1793 the turbulence of the Revolution drove him to a refuge at Melun, where he was appointed pharmacist in the military hospital. It was in this way, perhaps, that he escaped the fate allotted to Le Blanc and to Lavoisier; for, as the latter was rudely informed, the directors of public misfortune believed that there was no longer use for savants. He was recalled to Paris, in the year following, as adjunct professor of chemistry in that foundation of learning which, in 1795, became that École Polytechnique, now so well known to all of us.

Vauquelin was an industrious chemist from the time when he entered Fourcroy's shop to the time of his death, at the age of fifty-six, full of honors. He published much before being called to teach in the École Centrale, at the age of thirty-one years; but it was after this that his fertile brain did so much to make chemistry the French science which then it undoubtedly was. His earnest and honest soul found at last its home in the laboratories of the Polytechnique, and it was here that he again took up the work with which his medical friend, Macquart, and himself had employed themselves in 1789. This time he was rewarded with success.

An account of this work is to be found in *Annales de Chimie*, 30 *Nivose*, *an* VI. (19 Janvier, 1798), vol. xxv., Paris, an VI. (1798). It is printed on page 21, under the title: *Mémoire sur une nouvelle substance métallique contenue dans le plomb rouge de Sibérie, et qu'on propose d'appeler Chrôme, à cause de la propriété qu'il a de colorer les combinaisons où il entre. Par le cit. Vauquelin. Lu à la première classe de l'Institut nationale, le* 11 *Brumaire, an VI.* The author recalls the work previously done by Macquart and himself, and adds that since that time Bindheim had stated he had found many bodies in the mineral, including molybdic acid. Then comes a statement of Vauquelin's work, and it is conclusive that he had surely identified some of the oxides of chromium. The same volume (p. 194) has Vauquelin's second *mémoire*, and in this is detailed the method by which he isolated the metal, the character of which is described.

The date given as that of the reading of the first paper, *le* 11 *Brumaire, an VI.*, corresponds to November 4, 1797, and this must be assigned as the date of Vauquelin's discovery of chro-

mium. It is true that he did not isolate the metal until a subsequent time.

Two dates are given for the discovery of chromium, 1794 and 1797. This discrepancy has grown out of the fact that nearly all French scientific periodicals were suspended from 1793 to 1798. When they resumed, some of them were antedated for as many as four years. One should be especially careful about the dates of the *Journal de Physique*, *Annales de Chimie*, and *Journal des Mines*, and should critically examine into the dates of almost any other French print purporting to have been published during the interim above indicated.

During the latter part of the last century the mineral crocoite (Dana) was an object of attack from many chemical workers, because it was believed to contain a body not then known. Bindheim had declared that this peculiar body was molybdic acid; but his statement was by no means concurred in. Klaproth took up the problem, and in the right way; but his results were anticipated by the discovery of Vauquelin. Publication of Klaproth's work was made in a letter which the editor of Crell's *Annalen** had asked of him, and which stated that he had dissolved Siberian red lead-ore in hydrochloric acid, and after freeing the solution from lead chloride, had saturated it with sodium carbonate; whereupon, "*der Metallkalk fiel mit bläulicher, dem Spangrün sich nähernder Farbe.*" A small sample of this metallic calx (which was, doubtless, chromic oxide) he enclosed in his letter. He adds that he had fused it on charcoal, with microcosmic salt and with borax, and he gives the characteristics of the beads resulting.

Further notice of his work is to be found in his chemical dictionary,† under the caption "Chromium." It is but a brief statement of the facts that Klaproth had suspected the presence of a "new metallic substance" in the lead-ore, and that Vauquelin had anticipated his work. Curiously enough, Klaproth makes no mention of the matter in his *Beiträge*."‡

* *Chemische Annalen*, Helmstadt, 1798, vol i., p. 80.

† Klaproth and Wolff, *Chemisches Wörterbuch*, Berlin, 1807.

‡ *Chemische Beiträge*, 5 vols., 1795–1810.

While prosecuting again my research into the early history of chromium, I have received courteous and friendly aid from the librarians of Yale University (Addison Van Name), of Columbia College (George H. Baker), and of Lehigh University (W. H. Chandler), as well as from the Secretary of the Institute; and

Subsequently (1798) the element was found in another mineral from the Urals, to which was given the name of chromic iron, a mineral so abundant that its chromium might be extracted for use as coloring matter to be applied in the arts. By fusing in a pot a mixture of the powdered mineral and potash nitrate, the oxide was converted easily to chromic acid, and finally into potash chromate, which, in a comparatively pure state, could be separated by water from those parts of the ore not decomposed in the fusion. The next step was to treat the yellow solution with a mineral acid, whereby the potash chromate was compelled to yield to the acid one half its potash, forming potash bichromate, which was readily separated by crystallization from the mother liquor.

In 1820 Köchlin learned that chromium salts were well suited to dyeing Turkey-red. Shortly afterwards they were found to be adapted to the production of several bright colors in textile fabrics and in painting, especially in the dyeing of wool and the decoration of porcelain. But the excessive cost of the salts restricted their economic use. Potash nitrate was costly, as were the ore and its reduction to powder. Moreover, there was but partial oxidation of the chromic oxide present in the ore fused, and the fusion-pots were costly. But success offered high reward. Cheaper potash carbonate was in part substituted for the costly nitrate, and fusion was accomplished on the hearth of a reverberatory furnace instead of in pots. Next it was learned that when pulverized chromic iron was heated in a bath of potash carbonate held on the hearth of a reverberatory, its oxide could be converted to chromic acid simply by means of the oxygen contained in hot air. And the problem seemed solved, because hot air in abundance could be supplied from the combustion-chamber of the furnace itself. But there were those who saw then, as now we see more clearly, that no problem in chemical technology ever reaches its final expression. Potash carbonate was not only costly, but in a hot furnace it was volatile. Moreover, the oxidation of the charge held on the furnace-hearth was incomplete, because the fused

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the hospitality of the Boston Public Library, the Library of the Massachusetts Institute of Technology, the Peabody Library and that of the Johns Hopkins University has been freely accorded to me. I beg to make my grateful acknowledgments.

alkali protected the ore from hot oxygen passing over it; and this remained true, no matter how industriously the charge were stirred about. There came a mighty advance in the industry when, in 1845, Stromeyer introduced slaked lime into the furnace-charge. By means of this infusible body the charge of ore and alkali could be maintained in a sponge, even in a furnace heated to bright yellow; and through the spongy charge an oxidizing-flame readily passed. From that time onward it has been possible in this way fairly well to convert chromic oxide to chromic acid. And, in addition thereto, potash-lime proved far superior to the simple alkali as a means of decomposing chromic iron.

In the present state of our knowledge, for the production of alkali chromates there is made a mixture of pulverized chromic iron with lime which has been slaked by a solution of potash carbonate. This is subjected upon the hearth of a reverberatory furnace, to an oxidizing flame, which converts to chromic acid the oxide resulting from the decomposition of the ore. Finding itself in the midst of alkali, it at once unites therewith, forming potash chromate, which is not decomposed under the conditions present.

In 1883 a German manufacturer substituted soda carbonate for the potash carbonate exclusively used before, and in that way produced sodium chromate. Otherwise, there has been no published notice of any material advance in the manufacture of chromic acid salts since the days of Stromeyer.

At present, all establishments in this business produce bichromates of both potash and soda, the chief consumption of which is in the dye-houses of manufacturers of textiles. Nearly all black and brown fabrics, as well as most buffs and some reds and yellows, are colored with their aid. Perhaps tanners of leather stand next as consumers, being followed by makers of colors used in painting and in many kinds of printing. Of the very many other forms of consumption of alkali bichromates, the manufacturer of them knows but little.

Previous to 1827 the production of chromic acid salts was limited. The methods of manufacture were crude, as has been said. The supply of chromic iron was limited, while its cost was oppressive. I have not been able to learn much of the sources of early supply; but tradition relates that they lay

chiefly in the Urals, not far removed from Ekaterinburg, and in the drainage-basin of the streams which finally find their way to the Arctic Ocean through the river Obi. The ores were loaded upon rafts, and thus floated to navigable waters, after a voyage lasting often through two summers. Thence, they found their way to the Arctic, and at last were landed in western Europe. But in the year named, discoveries of chromic iron made in this country entirely altered the status of the industry, and introduced the period of abundance which we still enjoy. From 1827 to about the beginning of our Civil War, the Baltimore region supplied practically the world's consumption of chrome-ore.

I propose to write of the chrome industry of the southern part of the Appalachians, a region described by Mr. Willis* as that

> "Belt of disturbed Paleozoic strata which extends from New York, through Pennsylvania, the Virginias and Tennessee, to Georgia and Alabama.
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> "This is an area of about 60,000 square miles, 900 long and 50 to 125 miles wide. It is a geologic province distinguished by the age of its strata from the region on its east, and by the facts of its structure from the horizontal rocks on the west. Toward the east extend crystalline rocks much older than the Paleozoic and part of that continent which yielded the materials for Paleozoic sediments. On the west is the area over which the mediterranean sea of North America prevailed during the periods from Cambrian to Carboniferous. Between the continental edge and the open sea was the narrow belt where mechanical and organic sediments accumulated in great bulk. This strip is the zone of strongly developed structural deformation."

Together with the "disturbed Paleozoic strata" which Mr. Willis has regarded as the Appalachian region of his study, this paper will include the "crystalline rocks much older than the Paleozoic," which lie between it and the present coastal plain. In a consideration of economic mineral characters, the two regions cannot be so sharply divided as they may be for precise geologic study.

It may be regarded as appropriate, at this meeting in Atlanta, to recall that the region we are now considering was once comprised in those parts of north Georgia and north Alabama which formed the home of the Apalache Indians. The sonorous derivative of their name has since been applied to the

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* "Mechanics of Appalachian Structure," *U. S. Geol. Sur.*, 1892, p. 217.

mountain ranges continuing for 800 miles to the northeastward of the country in which they dwelt.

In the summer of 1827, Isaac Tyson, Jr., saw in Belaire market, at Baltimore, a cart containing a cider barrel held from rolling about by means of some heavy black stones. They were chromic iron; and the knowledge necessary to identify them was at that time peculiar, perhaps, to the single American who saw them. He had made a study of them in their first-known American locality, that of the Bare Hills, near his father's residence, then six miles north of the city. He had studied "chrôme" in its primitive French literature, and had possessed himself of about all then known of the subject.

The stones in the cart of the countryman had been taken from the natural surface of a farm in Harford county, 27 miles northeastward from the city. This farm immediately passed into the possession of Mr. Tyson. Its surface was comparatively flat, but sufficiently diversified to afford ample drainage. The more northerly strip of it gave promise of generous agricultural importance; but the greater area was covered with woodland in strange contrast to that portion. The forest trees were of stunted growth, displaying a poverty of limbs and of foliage which indicated a deficient supply of natural sustenance. Of grass and of herbage there existed but little; and, in consequence, the face of the land was scarred with paths formed by running waters. Apparently the ground offered so little resistance to erosion that each heavy rain had left its visible record in the shallow channels it had dug. At intervals were to be seen groups of scattered blackish stones, each of which examination showed to be honeycombed, and interlaced with thin hard ribs of siliceous rocks. Underlying the shallow covering of earth, the rocks were serpentine, which seemed to occupy a belt a mile wide, and to extend both northward and southward beyond the reach of vision. A few hundred feet from the southerly edge of the fertile land, in the midst of the serpentine, were scattered the blackish stones from among which had been taken the heavy pieces wherewith to steady the cider-barrel in the farmer's cart. Scattered thickly over the surface in the midst of an impoverished woodland scene, covering the figure of a rude circle a hundred feet in diameter, there lay nearly thirty tons of stones of chromic iron, which then

formed the total chrome-ore supply known to exist on this continent. And though they had been regarded always as a mere incumbrance on land almost worthless, their value was much beyond that of the entire Reed farm besides.

The ore which lay upon the surface was carefully collected and carted to Baltimore, and to a building which stood beside the Back Basin, on what is now Caroline street. The site is occupied at present by the depot established by the Adams express company for the receipts of shipments from the packers of canned edible materials. Here it was put into barrels and shipped to Liverpool, and finally to the establishment of J. & J. White, at Glasgow, where were produced citric and tartaric acids, soda carbonate, and bicarbonate, as well as potash bichromate, of which the works were then important producers.

The Reed mine had no outcrop of ore in place. We now know that the ore which lay upon and within the soil was what remained of a pocket which once had existed above the later natural surface. In the form of rounded stones of various magnitudes, the weather-resisting chromic iron had maintained its integrity, while the readily-yielding serpentine had been broken down and washed away. Or, if the mine had an outcrop it was of but a mere string of ore. The step pursued was the one invariable in those days, and not unknown in these, namely, to sink a shaft where the ore ought to be—whether it were there or not. By mere good fortune, at the depth of 8 feet, the shaft struck an ore-pocket which dipped westward 75 degrees under the horizon. The deposit was ultimately found to be 80 feet in length, 25 feet in width, and 4 to 8 feet in thickness. In addition thereto, near by, two other pockets of smaller dimensions were found.

The owner of the Reed mine was at no loss as to the course he ought to pursue. His knowledge had pointed out to him that chrome-ore was found, so far as he was informed, in serpentine rocks only; and his trained habits of observation had taught him that exposures of serpentine could be recognized in a landscape, even at a distance. As yet, he had a monopoly of such information; and it gained for him a monopoly of the chrome-industry in America.

His next discovery was of chromic iron in the form of grains of sand, in the beds of brooks on a tract of land called

"Soldiers' Delight," an estate which lay 16 miles northwestward from Baltimore. He bought all the territory which his examinations had shown to be valuable, and then contrived a simple apparatus for separating the ore from the specifically lighter materials with which it was mixed.

This variety of ore, because of its form and its associations, is called "sand-ore." It is a result of the disintegration of the serpentine rocks in which it has its home. The waters of rain and of melting snow move the ore-particles along the hillsides, and finally bring them to the brooks; here they remain because of their gravity, while the lighter constituents of the broken-down serpentine are washed away. After this manner, there has been deposited more or less of sand-ore in most of the brooks lying in the serpentine fields; but few, indeed, of these brooks have any commercial value.

Sand-chrome streams, however, may afford mines of which the values do not vanish. Even while the ore is being removed from a fruitful brook-bed, the waters which fall upon the land are replenishing it. Experience has shown that where a brook lies within a rich territory, it can be worked profitably at intervals of about fifteen years. The reader will observe that here we have nearly an inexhaustible mine, a form of profitable ground, which many of us have believed to exist only in the hopeful imagination of the ultra-sanguine.

When of equal percentages of chromic oxide, sand-chrome is as valuable as rock-ore—often more valuable. That from Soldiers' Delight was packed in barrels and carted to the storehouse in Baltimore, whence it was shipped abroad with consignments of rock-ore.

In 1828, Mr. Tyson saw on the Wood farm, in Lancaster county, Pennsylvania, a group of stones of chromic-iron which far exceeded in quantity that which lay upon the surface of his Reed mine. He at once leased the ore-right of the farm, and acquired the fee-simple in 1832. This surface-ore was collected and carted 12 miles to Port Deposit, from which point there is water transport to Baltimore. This was the Wood Pit (in literature, the Wood mine), a deposit which, for extent and scientific interest continues to be the most famous the world has as yet produced. For a condensed description of it, we cannot do

better than copy from a volume published by Professor Frazer,* fifty-two years after its discovery:

"The Wood pit was opened in 1828, and has worked to the present time, except from 1868 to 1873. The total output has been about 95,000 tons of chrome-iron ore. Of late years, but a small force has been employed, producing 500 tons to 600 tons of ore yearly. The assay value varies, as with other kinds of ore, but within smaller limits.

"The country-rock is serpentine. The ore-body, as proven, is almost 50 fathoms long at its greatest extension. Depth proved to 120 fathoms. Pitch of the mine is from 40 degrees to 60 degrees under the horizon. The strike is nearly east and west at the outcrop, and nearly north and south on lower levels. The width of the ore-bearing rocks is from 10 to 35 feet, or may be taken generally at 20 feet. In this space occur the chrome-ore and gangue (mostly serpentine), which show a general attempt at stratification, conforming to strike and dip of the mine. But occasionally, a branch of ore will stand vertically and extend itself into the foot-wall; or it may be horizontal, and do the same thing. The mine is worked by ordinary 10-fathom levels, winzes, etc.

"This mine is famous throughout the civilized world for specimens of minerals which it has furnished to all cabinets. . . . . The ore is thrown out almost pure, and without admixture of gangue. . . . . About 5 per cent. of the ore is crushed and washed, the remainder being pure enough to ship without washing. . . . The serpentine which forms the country-rock here is unstratified, and is about ¾ of a mile (or 1.3 kilometer) in breadth. The strike of the vein is about W. 12 degrees S. . . . . The sandy chloritic slates to the north of the mine dip S.—50 degrees."

In connection with the last sentence, the writer begs to observe, that the pitch of these slates, as determined by Professor Frazer, is very nearly that of the ore-body near by, in the serpentine rocks. The fact is suggestive.

The group of three mines near Rock Springs, in Maryland, came next in order of discovery. In literature they are called "Texas," in deference to a village 3 miles distant. They are the Jenkin's pit, the Lowe pit and the Line pit, the latter of which was begun in Cecil county, Maryland, and because of its northward pitch, soon ran across the State boundary and into Lancaster county, Pennsylvania. The mine in Maryland was owned by Mr. Tyson, who paid to the proprietors a royalty for so much of the ore as lay under the soil in Pennsylvania.

Two hundred yards east of the mine stands a sandstone monument which protrudes 4 feet, perhaps, above the natural surface, and which is in section about 16 inches square. On its south front is sunken, in relief, the coat of arms of Lord Bal-

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* *Second Geol. Sur. Penn.*, CCC., p. 192, 1880.

timore, the Proprietary of Maryland; and similarly on its north front, there is engraven the coat of arms of Penn, Lord Proprietary of Pennsylvania. It is called the "third line-stone," and it marks the end of the 15th mile, measured from the beginning of the tangent (at the northeast corner of Maryland) which Mason and Dixon established as the line which was to be the boundary between the lands of Penn and those of Baltimore. The monument had been cut in England and was one of those which Mason and Dixon planted in 1764, at the ends of each five miles of the tangent they established.

Having at hand a monument so full of authority, one from which there was no appeal, it was not difficult to mark on the walls of the sharply pitching mine many points at which its cavity intersected the plane of Mason and Dixon's line. There has never been a controversy upon what ore royalty should be paid by Mr. Tyson to his neighbors.

The serpentine belt, passing northeastward through Harford county, Maryland, crosses the Susquehanna at the Bald Friars, and, 4 miles beyond, encounters Mason and Dixon's line. Then it alters its course to due east; passes the Line pit; 4 miles beyond, it passes the Wood pit; and 10 miles further east, it arrives at the Boone farm. There it reaches the terminus of Mr. Tyson's ownership of land, which is equivalent to saying, the terminus of profitable chrome-ore territory. All the brooks of the serpentine lands, from Wood pit 10 miles east to Boone farm, contained sand-chrome ore. Many of them were, and are, valuable. The single rock-ore mine in all this 10 miles' length of territory, has not been, and does not now seem, important. And no valuable chrome-ore deposit has as yet been found to the north or the east of the Boone farm. (No account is here taken of the Canadian mines).

To the southward of Baltimore, the range of serpentine exposures may be traced across Howard county, through Montgomery and across the Potomac, 18 miles above Washington, where they enter Loudon county, Virginia. All of this region was diligently searched by Mr. Tyson, and in none of it did he find any profitable chrome-ore ground. It was for this reason that his ownership of land had its southern extremity at Soldier's Delight, northwest of the city. From that point to the northern extremity at the Boone farm, is, by county-road,

about 60 miles. Within that scope of country lay all the valuable chrome-ore territory then, or now, known to exist in Pennsylvania, Maryland or Virginia.

The owner of this chromium territory came to believe that he could not indefinitely continue to enjoy his monopoly. He was more and more of opinion that other regions would be found which were as rich as his own, and that he would not be able to control the whole of them. There would grow up a competition which would depreciate the value of his ores, in addition to invading their market and thus restricting the volume of their sale. It was for this reason that he felt impelled to establish a factory for the consumption of the surplus of his own ores. This he did, in the year 1845, on land lying on the Back Basin, near to his chrome-ore depot in Baltimore.

In connection with this mention of the Baltimore factory, there are good reasons for believing that the subjoined passage will find welcome among the members of the Institute. It is taken from page 122 of a quarto volume* of more than 500 pages, prepared for the State by "Members of the Johns Hopkins University and Others."

"Among the first steps of Isaac Tyson, Jr, was to apply, in 1846, to Yale College for a chemist for his [chrome] works. In response, a young man named W. P. Blake, who was then a student in the chemical laboratory, was sent. For a while Mr. Blake did excellent service in the new factory, but he was not willing to remain.

"Mr. (now Professor) Blake was the first chemist to be employed in technology upon this continent; while the Baltimore chrome-works were the first to appreciate the value of chemistry. After the departure of Mr. Blake, another chemist was secured from the first laboratory ever instituted for the teaching of chemistry; that founded at Giessen by Liebig [in 1830]. In succession came another chemist, from the same laboratory, and this gentleman is yet employed in the works."

It is now 49 years since this "young man named W. P. Blake," set up his laboratory in the Baltimore Chrome Works; 49 years of industrious teaching and working. And he is today one of the most fertile and voluminous teachers whom this Institute enjoys, and one among those most welcomed.

The judgment, in his field, of the owner of the American chrome-industry, was indeed rarely at fault. Even while he was perfecting his factory, there were being laid the founda-

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* *Maryland: Its Resources, Industries and Institutions*, Baltimore, 1893.

tions of the conditions he already had foreseen. In the year 1846, Dr. J. Lawrence Smith, another American, who had studied chrome in the Baltimore region, was called into the service of the Turkish government for the purpose of studying the mineral wealth of its empire. During the following four years, he made discoveries which transferred the monopoly of chrome-ores from America to the eastern possessions of Turkey. The following brief statement of bare fact was printed by Dr. Day, on page 120 of his volume of *Mineral Resources of the United States* for the year 1888.*

"The principal supply of ore [chrome] for the world comes at present from Asia Minor. It was discovered there by Prof. J. Lawrence Smith, when making mineralogical explorations for the Porte in 1848. He found it on a journey south from Brusa, a town 57 miles south-southeast from Constantinople. Near Harmanjick, 10 or 15 miles farther south, he found another abundant deposit. There is another field near Antioch. He describes the ore as in masses rather than in veins. The present supply comes from deposits somewhat farther south. It is taken on the backs of camels through a mountainous region to the seaports of Macri, near Smyrna, and Ghemlek, near Brusa."

For reasons which need not now detain us, the Turkish deposits were but slowly developed, and it was not until ten years afterwards that their full importance was felt in America. Since that period they have dominated the markets of the world, and for many years they have supplied the Baltimore factory itself. They have made of no avail the ores of California, of the Urals and of the Danubian provinces, and they have seriously hindered the production of those of New South Wales and of Norway. Those of the Baltimore region have become private property, held in reserve.

In Virginia the serpentine exposures of the crystalline rocks may be found in small fields scattered through the Piedmont country. The city of Lynchburg is in part built upon one of them. It is altogether likely that they might be traced, in similar position, to the southern end of the crystallines in Georgia. While in search of something else, the writer happened upon one of them near Washington, in that State; but so far as known, all fields occurring in this horizon, southward of the Potomac, are comparatively small, and are not known to contain any valuable quantity of chrome-ores. There is held

* *U. S. Geol. Sur.*, Washington.

out but little hope of reward for him who values them for that mineral, and hence they have not received a really critical examination. So far as is at present known to me, good deposits of chromic iron do not occur in small fields of serpentine or in fields as yet not completely altered to serpentine.

Within the region of our study, all the chrome-deposits north of the Potomac lie in the ancient crystalline rocks east of the mountain chain. Those south of the Potomac lie in the midst of the Appalachian region, in the valleys among the folded mountains themselves.

We may regard the region of promise as beginning with the deposit on Jack's creek, in Yancey county, North Carolina—a deposit known to all those who have experienced even the slightest interest in the industry. To the southwestward, through Madison, Buncombe and Haywood counties—a distance of 60 miles—many discoveries have been made of late, concerning which I am not well informed. Further southward lies the deposit near the church in the Balsam Gap. Southwestward still, near four hours' walk, lies Webster, the county seat of Jackson, in the village street of which may be seen a deposit of chromic iron of no little importance. Following along the mountain ranges, even to their end in northern Alabama, one sees evidences of chrome-ore in many of the serpentine exposures encountered; but as yet not a deposit of them all, within this mountain region, has been carefully explored. There has been lacking incentive or technical equipment necessary to the task. It is a region of lofty mountains, of swift-running, clear waters—the grandest of all the Appalachian realm. But relentless trade has not yet smiled upon it.

The village of Webster has to offer that which is rare in the region of our study—a deposit of nickel-ore in serpentine rocks.

It was first observed (by me) in the early summer of 1878, at a point on the hillside in the eastern edge of the village, a space from which running waters had washed away the thin covering of soil. The greenish ore was to be seen in the joints of the rocks, in thin sheets, which could be detached with a knife-blade. I regret to say that I have lost my notes of this deposit, including my analysis of the nickel-ore; but I believe it was a hydrous silicate of nickel containing magnesia. No determination was made of the quantity of nickel contained in

a given volume of rock, because, in the then state of our knowledge, the proportion did not seem sufficient in any event to warrant extraction. A more extended examination of the locality gave evidence that some acres of the soil were underlain by this peculiar deposit of ore.

It is not unusual to find nickel associated with chromic iron. It occurs at the Wood pit in the form of genthite, a greenish mineral incrusting chromite and deposited in the joints of the mine wall-rock. Similarly, it is found in the mines of the Line Pit group. But the occurrence of the mineral affords clear evidence that its presence is accidental; that it had been deposited from watery solution subsequent to the formation of its enclosing rocks, and that it had been concentrated, by some manner of chemical action, from those rocks.

In writing of the "third (or Green Mountain) series" of aluminous rocks of the Laurentian system of eastern North America, Sterry Hunt says:*

> "The great predominance of magnesia in the forms of dolomite, magnesite, steatite and serpentine is also characteristic of portions of this series. The latter, which forms great beds (ophiolites), is marked by the almost constant presence of small portions of the oxides of chrome and nickel. These metals are also common in the other magnesian rocks of the series."

In other parts of the volume, he repeats the statement that it is not uncommon to find oxide of nickel present in basic magnesian rocks. The subject is further discussed by him elsewhere,† in a way which leaves no doubt that his observations were conclusive that nickel was often diffused through great masses of serpentine. Concentration of the metal in joints of the rocks, as at Webster, is but a natural result of the conditions upon which Dr. Hunt insists.

We now are familiar with the chromic iron deposits of the eastern part of America, from the Gulf of Mexico to the St. Lawrence; with the mines of California, from San Luis Obispo to the Washington boundary; with those of Norway, and of the Urals; with the deposits of the Danubian Provinces, of Asiatic Turkey and of Syria; and with those of the Gundagia-Tumut district of New South Wales. With this knowledge before us,

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* *Chem. and Geol. Essays*, 1875, p. 32.

† *Geol. of Canada*, 1863, p. 738.

we can say distinctly that, wherever found at all, chromic iron is found in serpentine rocks. This is a most peculiar condition, and leads to the inference that a study of the genesis of serpentine ought to lead us to a knowledge of its cause; as the work of Dr. Hunt has taught us why it is that nickel is at times found concentrated in the joints of certain of the basic magnesian rocks.

Just beyond the westward limits of Baltimore, abundant masses of a dark-colored, heavy rock are to be seen upon the natural surface. Fifteen miles to the northeast of the city, masses of the same character are encountered again; and they may be followed, still northeast, through Belaire and Darlington, to the Susquehanna at Conowingo. Beyond the river, they continue in the same direction, to Mason and Dixon's line near the Line Pit already mentioned: henceforth, they run due east, with the line, to its tangent-point, and onward into Delaware. These masses represent the weather-resisting rocks that underlie the surface upon which they are found.

Until recently, it was not known what kind of rocks this bed contained. A skilled geologist has called them in one place hornblende-gneiss, and at another point has regarded them as trap. In some localities they might be mistaken for syenite; and into that error the writer once fell.*

Under the leadership of the late George H. Williams, whose untimely and almost irreparable loss we still deplore, there grew up in the Johns Hopkins University a group of geologic workers who have solved many of the difficult problems offered by the rocks in Maryland. The learned Professor himself attacked that of the "nigger-head" rock just mentioned, and published his work under the title: "The Gabbros and Associated Hornblende Rocks Occurring in the Neighborhood of Baltimore, Maryland."† As the title indicates, this long-extended dike, or rock-bed, consists of ancient lavas, to one member of which Chapter V. of the *Bulletin* is devoted. The following summary of it was made by its author:

"Chapter V. contains descriptions of the third or olivinitic rock type, which is abundantly associated with the gabbro and gabbro-diorite. These rocks are younger than the gabbro, since they break through it in the form of dikes. They

* *Second Geol. Sur. Penn.*, CCC., p. 195.

† *U. S. Geol. Sur.*, *Bulletin*, No. 28, p. 12, 1886.

are rarely rich in feldspar, and are sometimes altogether free from it. By a gradual loss of olivine, they seem also to grade into another massive rock, composed almost wholly of diallage and hypersthene. These masses are all very rich in magnesia, and readily give rise to the serpentinous hornblendic and talcose rocks which are everywhere abundant in and about the Baltimore gabbro area. As far as could be observed, the olivine seems always to form serpentine; and the pyroxene (no matter what be its form), hornblende. This latter mineral often suffers a further alteration to talc, its lime separating out in the form of calcite. In this manner the hornblende serpentines, like those of the Bare Hills, have been derived from original eruptive aggregates of olivine and bronzite, similar in character to many rocks which are still to be found near them in a quite unaltered condition."

The author gives minute descriptions of these rocks, with detailed analyses, both microscopic and chemical; but at no point does he mention chromic iron as a constituent of the mother-rock of the serpentines. It is scarcely conceivable that a writer so careful and so thoroughly equipped would have failed in such mention had the mineral been anywhere present.

Frederick D. Chester, once a worker in Prof. Williams's laboratory, has studied the gabbro area lying along Mason and Dixon's line. His work on "The State Line Serpentine and Associated Rocks"* is minute in detail; and as regards the genesis of the serpentines, it is entirely corroborative of the work done by Prof. Williams upon the same rocks, forty miles further to the southward. At no place does he mention chromic iron as a constituent of the rocks from which these serpentines were metamorphosed.

Many other writers have studied the genesis of serpentine, and those who have done their work with the microscope are in general accord; but so far as I have been able to discover, none of them have, as yet, traced the chromic oxide, nearly always present, from the matrix to the serpentine.

There are, indeed, few who have displayed in their writings so wide a view of geologic knowledge, and so broad an acquaintance with the work of others in that field, as did Sterry Hunt. It is for this reason that I have diligently searched what he has left us, feeling assured that if anything were known of the genesis of chromic iron deposits, he would at least refer to it. His *Mineral Physiology* has but two references

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* *Second Geol. Sur. of Penn.*, *Annual Report* for 1887, p. 93.

to the subject, neither of which is here of service. His *Chemical and Geological Essays* contain 11 references, all of which concern themselves with the occurrence of chromium in basic magnesian rocks, of which serpentine is the type. Regarding the genesis of deposits of that metal, all that he has to offer is the following passage from pages 237 and 238 of that work:

"The views maintained by Lieber, Wurtz, Genth and Selwyn as to the solution and redeposition of gold in modern alluvial deposits seem to be well grounded, and we are led to the conclusion that the circulation of this metal in nature is as easily effected as that of iron or of copper. The transfer of certain other elements, such as titanium, chrome and tin—or, at least, their accumulation in concentrated forms—appears, on the contrary, to require conditions which are no longer operative, at least at the surface of the earth."

Modern methods of petrographic work with the microscope, such as that of Prof. Williams and of Frederick D. Chester, lead one to suppose that, so far as concerns the genesis of serpentine, we must give up much of our former beliefs; that often, if not always, it is metamorphosed from certain forms of ancient lavas, seems more than probable. It is to workers in this field that we ought to look for knowledge as to the source from which, and the manner in which chromium minerals are derived, and the agency by which they have been deposited in such bodies as we call mines. It is an impressive fact that chromic iron is found in serpentine only, or in some rock nearly akin to it, and, like it, metamorphic. And yet, those who know most of the subject lead us to suppose that it does not occur in the rocks out of which its only matrix (as known to us) is generated. Whence comes chromic iron, how it gets into serpentine (which is a secondary rock), and by what agency it is concentrated in mines, we are not, as yet, informed. Measured in terms of human agency, and the duration of a lifetime, it is one of the most insoluble, most resisting of the forms of matter known to chemistry. And we are ready to assert, as did Sterry Hunt, that its movement within the rocks, and concentration in them, seem to require conditions which we do not now recognize as operative within the earth's crust.

Chromic iron occurs in masses such as miners call pockets; that is, in masses having no definite form and no tendency to extend themselves in a special direction. But this is a rule

having many exceptions; for frequently a pocket will lie upon a foot-wall having a definite form and a persistent pitch, as at the Reed mine and the Line pit. Usually, pockets extend in depth to from 20 feet to 40 feet when they have no well-defined foot-wall; when they have one, they may be expected to reach to three times that depth. In the Urals the deepest pockets reach to about 20 meters (as stated to me by Prof. Nicholski, of the Mining School at St. Petersburg), and they may be taken as typical of the European and Asiatic mines generally.

The Wood pit stands alone among chrome-mines. It is as truly an ore-chute, a vein, as are the copper-mines in the crystalline rocks at Capelton, in the province of Quebec, or at Ely or at Elizabeth, in Vermont. It has a perfectly well-defined and smooth foot-wall pitching about 60 degrees for 300 feet, and about 45 degrees for 400 feet, at which depth is its present bottom. There is this peculiarity; the lower 400 feet of it is a warped surface, which bends, through a quadrant, to the left, as one looks down the incline. From this condition, it is fair to suppose that the ore was not laid down upon the floor of a lake or a pond, as were certain other iron-ores, and this inference is sustained by the outline of chrome-pockets in general.

The Form of Fissure-Walls, as Affected by Sub-Fissuring, and by the Flow of Rocks.

BY WILLIAM GLENN, BALTIMORE, MD.

(Atlanta Meeting, October, 1895.)

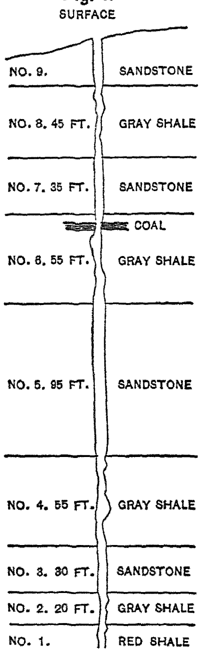
THE Ritchie vein, of Ritchie county, W. Va.,* was a straight fissure, about 3600 feet in length, which cut vertically downward across the horizontal beds of shale and of sandstone to a depth not ascertained. It might be described as lying in a vertical plane, bearing 12 degrees north of west. The geological horizon is that of the "upper barren coal measures" of the Appalachian coal-field, being a part of the strata lying above the Pittsburgh bed. The vein matter was declared by Prof.

* See *Trans.*, xxiv., 195.

Lesley* to be a form of asphalt, and was called grahamite by H. Wurtz subsequently.†

The configuration of the east end of the vein was not ascertained precisely, but its west end might be called wedge-formed: that is, it narrowed symmetrically for 300 feet and then ceased. The vein-width varied with the nature of the enclosing walls; in the massive sandstone No. 5, Fig. 1, it was more than four feet; in sandstone No. 3 it was about 3 feet. In passing through shale beds, its width was various and exceedingly irregular, perhaps never less than 3 inches and rarely more than 3 feet. It was most noticeable that the sandstone side-walls were vertical plane surfaces, or nearly so, while the shale side-walls were rough and more or less contorted.

Fig. 1.



VERTICAL CROSS-SECTION OF THE RITCHIE VEIN: CLEFT EXAGGERATED

The surfaces of the sandstone walls had not the texture of a freshly split sandstone. They had not sufficiently the abrasive quality of a fresh sandstone surface, but rather that kind of smoothness which would have ensued had water run for some months, or some years, over the freshly split rock.

So far as might be judged, heavy pressure of the soft shale walls against the enclosed grahamite, had obliterated entirely any distinctive surface which may have been possessed by those walls at the time when grahamite entered the fissure.

Whatever may have been the connection of fissures with the origin of the mineral deposits of the Appalachian region, I am compelled to say, after an acquaintance of nearly forty years with the mines of that region, that the Ritchie is the only mine upon a true, simple fissure-vein which I have ever seen between the Canada line and the Gulf—that is, it is the only

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* *Proc. Am. Phil. Soc.*, ix., 185, 1863.

† *Am. Jour. Sci.*, II., xlii., 420, 1866.

mineral deposit enclosed in and confined to a simple fissure, crossing the stratification of the country. Moreover, the nature of the vein-filling is such as to eliminate from the problem involved in the study of its form the disturbing and obscuring elements due to chemical reactions between vein-matter or mineral-bearing solutions and the wall-rock. It was pointed out long ago, that igneous dikes, being fissures filled with material in fusion, are typical examples of fissure-veins—except that they are not economically valuable. This exception has an important scientific bearing, namely, that since such dikes are not mined, no opportunity is given for the detailed study of their walls in depth; and moreover, the effect of the molten material upon the country-rock has often been considerable, so that an exposure of the wall at a given point would not show precisely its form after the formation, and before the filling, of the fissure. In the case of a vein of such material as grahamite, neither the chemical composition nor the original temperature of which seems to have produced any discoverable effect upon its walls, we have, as it were, a practical preservation of those walls, protected against all metamorphosis, as they existed when the vein-material was introduced, and affected only, if at all, by subsequent movements of the rock itself. It should be remembered also, that the vein-material in this case was not deposited by slow incrustation or precipitation, but filled the mould at once. For the study, therefore, of the first period of the history of a fissure-vein, namely, that of the formation of the fissure itself, such a vein affords a unique opportunity; and if it has been, like the Ritchie, worked out, exposing the walls over a large area, its value to science in this respect is thereby doubled.

The only similar occurrence known to me is that of the (likewise exhausted and abandoned) Albert mine, of Albert county, Province of New Brunswick, the first account of which was contained in a privately-printed report written May, 1851, by Richard C. Taylor and Prof. James Robb. In the succeeding year, Mr. Taylor described it before the American Philosophical Society.* He states that the contained mineral is:

* *Proc. Am. Phil. Soc.*, v., 241.

"A remarkably pure and brilliant asphalte. It was at once apparent that the mineral substance of the Hillsborough or Albert mine occurs at a very high angle which varied from nearly perpendicular to within 10 to 20 degrees of verticality. Its position is in the midst of a formation which consists of highly bituminous calcareous or marley shale. The opposite sides or walls of the Hillsborough vein are very dissimilar at certain points, yet at intervals, for short distances, they are parallel and conformable."

A somewhat exhaustive description of this mine was printed in 1865 by Prof. Hitchcock.* He declares that the rocks there were of the Carboniferous age, and that they belong to the Acadian coal series. Continuing, he states that the lowest member of the group is the Albert shale, which outcrops at three points, in the center of the largest exposure of which was sunk the Albert shaft. He says:

"The greater portion of the shale will sustain a fire without the aid of other fuel. Other layers are more bony, and others still highly ferruginous."

The vein-walls crush mine-timbers and tend to close up—conditions which he ascribes to "the great plicating force acting from the direction of the ocean," which "crowded the Hillsborough rocks northwesterly." In support of this opinion, he writes:

"We might explain the falling of fragments by gravity, but not so easily the crushing of the timbers. The coal shows the effects of the crushing-process no less plainly. It is much broken, even to grains, and needs no pick for its removal from the vein. It will flow as easily as heaps of corn."

"The general course of the vein is N. 65 E., but the coal is frequently heaved southward by small faults. Its inclination is northwestward from 75 degrees to 80 degrees, often vertical. The body of the vein is extremely irregular, constantly expanding and contracting both laterally and vertically."

"The narrow portions of the coal are invariably contained in a harder rock."

As will readily be seen, I have quoted from the papers of Mr. Taylor and of Prof. Hitchcock only those sentences which have a bearing upon the subjects meant to be entertained in this paper.

One going into the Albert mine would have seen a vein of varying width, standing nearly vertical within rough walls of greasy black shale. He would have observed that the vein was compactly filled with a soft and highly lustrous black mineral, which crushed readily when rubbed between the fingers. It

† *Am. Jour. Sci.*, II., xxxix., 267.

was so fragile that it was not easy to get by any process a specimen half as large as one's fist. Apparently, exceeding pressure upon it had filled it with myriads of surfaces of slight adhesion and of no adhesion. Prof. Hitchcock, undoubtedly, was right as to the "crushing-process" to which the vein-matter had been subjected. The lustrous and pleasing albertite, together with the rough and dirty black shale walls, were left as prominent memories of a visit to the mine—a deposit which was to be regarded as singularly irregular in width along both the strike and the dip.

The Ritchie and the Albert veins were decidedly similar. Both of them contained forms of asphalt, both of them lay in the unaltered sedimentary rocks of coal-basins, and both of them were fissures in those rocks. The chief distinctions were these: Ritchie lay in alternating beds of shales and sandstones, well above the coal-bearing strata, while Albert lay in a massive shale bed, well below the coal-seams. We may entertain various opinions as to the genesis of these fissures, but necessarily we are compelled to imagine that they were a result of some manner of strain which at last fractured the rocks. Somewhat careful study of the clefts led to that conclusion, and offered no support to any other hypothesis whatever. At least, that was, and is, my own conclusion.

If their walls once were in contact, and were pulled asunder, why are they not perfectly conformable throughout? Why is it that they are now unlike, and why are they alike over certain areas, and over yet others nearly so? What has caused all these variations (and yet, in part, no variation) in the side-walls of the fissures? Why is the Albert vein so strangely irregular in width? And why, at Ritchie, is the fissure widest where it lies in a sandstone bed and invariably constricted where it lies between shale walls? Why do the walls of both fissures tend to move nearer each other, and to tighten upon timbers placed between them?

To consider, first, the Ritchie vein, it may be said that the heavy bed of whitish massive sandstone, No. 5, Fig. 1, in which the vein was best developed, lies just above water-level, through a region reaching far in all directions from the mine. It was from this bed that the stone was taken for the masonry on the railroad which connected the mine with the Baltimore and

Ohio road at Cairo, because the stone, while soft and easily worked, also was durable. Natural decay of the underlying shale had caused great masses of the rock to be precipitated from its outcrop into the narrow valleys through which the railroad passed. Blocks were easily blasted from such masses, and, being detached, were readily split with wedges into stones sufficiently determinate to be built, for the most part, at once into rough masonry for the support of railroad-bridges. Considerable quantities were so used.

If into a mass of such soft and homogeneous sandstone a series of holes be drilled in a straight line across it, and if steel wedges be placed in the holes and so driven as to maintain a nearly equal pressure by each wedge against the walls of the holes, the stone will split along a more or less plane surface, passing through the axes of all the holes. The wedges having been removed, the two parts may be brought together, and the junction will be a nearly perfect one.

When treated in this way, it may be imagined that the stone is ruptured along a potential surface of least resistance; for which reason only two masses should result. But in a stone such as mentioned, it is usual that additional pieces are produced. Thin slabs flake off from the two principal resulting masses and along the fracture-surface.

If the two chief masses of stone be now brought into the position which originally they occupied, omitting the flaked-off portions, evidently the junction will not be a perfect one. There will result vacancies which before were occupied by the flakes of stone not replaced. And if the two chief masses be moved somewhat asunder, it may be seen that their walls are not at all points coincident.

Examination of a newly-formed wall of shale, such as may be seen in deeper railway-cuttings, leads one to suppose that shale is decidedly homogeneous. A weathered exposure brings one to the reverse conclusion. For illustration, the great shale bed underlying the Subcarboniferous conglomerate in Pennsylvania, may be well studied along the valley which Wills creek has cut through Savage mountain, near Cumberland, Maryland. In the newly-made railway-cuttings, the reddish shale appears to have been formed solely from the finest of clay sediments, which fell in quiet waters. The rock seems singularly homo-

geneous. But the action of frosts and of running waters have written a different story upon the weathered and barren hill-sides of the valley. They have demonstrated that the rock-bed, here some thousands of feet thick, is built up of strata of various solubilities and of different resistances; of more or less pure strata of shale, interstratified with clayey sandstones, the layers having much diversity of thickness and indeterminate boundary-lines.

So far as my knowledge extends, such a structure is true of all heavy beds of shale. If a rock-bed of such various composition be pulled asunder, the fracture-surface necessarily must be an uneven one. The fact may be demonstrated readily by splitting a shale mass either with wedges or by the violent action of explosives.

The behavior of both sandstones and shales, when subjected to the action of a steadily augmented strain which finally ruptures them, has a bearing upon the study of both the Ritchie and the Albert veins.

A rupture occurring in alternating beds of shales and sandstones which lie nearly horizontal, ought to produce a fissure lying in or near a vertical plane, because such would be the position of the plane of least resistance. For the same reason, the fissure ought to be straight—a plane, and not a warped surface.

In a massive sandstone stratum having considerable thickness, the walls of such a cleft ought to be in part similar and in part dissimilar. If brought again into juxtaposition, they ought, for the greater part, to be in contact, but there should be surfaces which are not in contact. For we must suppose that the rock, when pulled asunder, was subjected to the same laws which govern it when split by wedges in a quarry. Flakes ought to have split off from the walls when the rupture occurred, and, having fallen away into the cavity, they ought to have left the vein walls dissimilar over parts of their surfaces.

That this was true at Ritchie, was shown by slabs of sandstone found at times as horses existing in the vein, and at other times as slabs, yet adhering in part to the sandstone mother-wall. They weighed variously from a few pounds to a few tons.

Chiefly because of a want of uniformity in the structure of

shale beds, even within the strata of such beds, a ruptured fissure in them would be likely to afford walls exhibiting warped surfaces, or, more probably and properly, zigzag surfaces, where the strata were of much thickness. I do not conceive that there would be any flaking-off from the walls, as has been observed of walls in homogeneous sandstone, because shale rocks are not given to the liberation of such forms as flakes or slabs not parallel with the bedding. But I think that there would be generated many small planes of weakness about the surface of fracture—planes which, either at once or subsequently, would bring about the liberation of masses of rock of greater or less size. In brief, the rupturing force may be conceived to shatter (or tear or loosen) the rocks, to a greater or less degree, along the surface of fracture.

I must frankly say that this opinion ought to be received with caution. I know of no supporting evidence which may be cited from the observations of others. It is, however, a result of many years of study made while removing shale rocks, and of observations in two fissure veins lying in part or entirely within shale beds.

It may be imagined that the two opposing side-walls of both the Ritchie and the Albert veins were unlike, because one of those walls had been displaced vertically, and that, if this had not occurred, they would precisely coincide now. Such an explanation is entirely reasonable, *a priori.*

With regard to the Albert mine, it is not possible for me to say positively that the side-walls had not moved vertically. It would be no easy matter to prove either that they had or that they had not. The entire mine lay within a greasy black shale-bed, having few distinguishing strata. My own brief study of the mine led me to believe, however, that there had been no vertical movement of either wall.

That one could see points where the two sides of the fissure came into contact, and that no evidence was disclosed at such a junction, that either wall had ever suffered displacement, does not finally dispose of the question. For such shale necessarily must weld, as do blocks of ice; and having come again into contact, the Albert walls soon would blend so as to show at such a junction no indication of what had occurred. But as both observation and inference are against any vertical displacement of

the walls, he who maintains that such a movement did occur, first must prove his proposition.

In the Ritchie vein there had been no faulting by vertical movement of either of the side-walls. After the removal of the grahamite, the empty fissure cutting downwards across the strata disclosed the rock-section perfectly. If the walls of the fissure were brought once more into contact, as at some future time they surely will be, the strata would be as if no displacement had ever occurred. It is true that there are no sharp lines dividing the horizontal strata of shale from those of sandstone. Two observers might differ as to precisely where lay the line dividing any two adjoining strata of those rocks. Therefore, it may be said, one of the vein-walls may have risen or sunken in some degree, and yet that condition could not have been detected by examination of the two side-walls at any point in the fissure.

But conclusive evidence was to be found in a 12-inch seam of grossly impure coal, mentioned by Prof. Fontaine,* as lying "near the top of gray shale No. 6." After the vein-matter had been removed, the narrow vertical fissure cutting downward across the horizontal coal-seam disclosed the fact that the divided seam lay in the two side-walls exactly in the same horizontal plane. It was thus proved that neither of the side-walls had been displaced vertically in any degree.

As will have been noticed, Prof. Hitchcock emphasizes his statement that the Albert vein-matter existed under severe pressure; that it was crushed "even to grains," so that it would "flow as easily as heaps of corn." He adds that the vein-walls crush mine-timbers, and tend to close up. Any one at all familiar with the conditions existing in and about the vein, will freely approve the spirit of the Professor's description. Undoubtedly, the vein-walls pressed heavily upon their contained vein-matter.

Similar conditions existed in the Ritchie cleft. There, too, vein-walls closed in upon mine-timbers placed horizontally between them. One could notice, almost from month to month, that such timbers were being more and more crushed at their ends. Although the rock strata were horizontal, the vertical

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* *Am. Jour. Sci.*; iii., VI., 409, 1873.

vein-walls surely were creeping in upon the cleft. Moreover, there was no escaping the evidence that the shale-walls were creeping faster than were those in the heavy sandstone bed; a condition which at first mystified us not a little.

In Prof. Fontaine's paper, already cited, he writes:

> "There is a considerable lateral pressure exerted on the mineral from the tendency of the walls to close in. As it is removed by the pick, the sudden yielding of the remaining portions is denoted by a series of pistol-like reports. The movement, however, is imperceptible."

It should here be said that the Professor's remarks apply more especially to the shale bed No. 2. They would apply almost as well to any other shale or thin sandstone beds. But in sandstone bed No. 5, pressure was not so severe upon the contained grahamite; at least it so seemed.

That all mine-walls tighten upon timbers placed between them, is a condition known to all who have been ordinarily observant underground. There is nothing unusual in the two cases we are now considering, only, that in them the movement seemed more than ordinarily pronounced. Consideration of it leads us into a field even yet not thoroughly explored. It brings us face to face with the subject of the flow of rocks under pressure, a study which may offer, as to one at least of the clefts herein described, an explanation of the phenomenon of varying width.

Passing westward from Cumberland, Maryland, the National road, after clearing the suburbs of the town, enters the passage which Wills creek has cut through the Fort mountain. The gorge is narrow and deep. It was cut vertically across the mountain axis. Standing in it, and looking at its precipitous northern side, one sees a veritable arch formed of heavy beds of whitish sandstone, supported by beds of red sandstone and shale, all similarly curved. The arch is about as long as the gorge, and as high as the crest of the mountain above the bed of the stream; that is to say, more than a mile long and a thousand feet high. Without breakage, without any loss of continuity, the heavy and stubborn sandstone bed was bent into a high and symmetrical arch. In the fewest possible words, the mountain is an anticlinal fold, which seems to rise out of the valley of the Potomac and to subside into the syncline which lies along the mountain's western flank.

Apparently, this great sandstone arch does not exist under duress, as does a bent bow. There are evidences that no internal stress exists within it. The arch is symmetrical, and although now relieved by erosion of the great load of rock formerly resting upon it, it retains its original form; a condition not likely to exist if active strains now affected it. The pressure which once necessarily must have acted to bend the mass, is no longer exerted. The myriads of groups of particles which constitute the arch have permanently altered their relative positions. Flow within the rock-mass has relieved internal stress—just as truly flow as is the movement of putty outward from between the approaching jaws of a vise.

In attempting to arrive at the extent of rock-flow which occurred in the Wills creek arch during the period of its elevation, we must not regard the rocks there as forming a single rigid mass. During the time of their movement, the various strata slipped one upon another, thus reducing the stress far below what it would have been for a single mass. Into how many sliding strata those rocks resolved themselves during the period of anticlinal movement, must remain a matter of conjecture; but it will be doing no great violence, if we regard 50 feet in thickness of the upper stratum of sandstone to have moved as one mass. Then, the upper limit of the visible bed now forms an arc of 5000 feet chord, and of 1000 feet versed sine, equal to an arc of 5517 feet length; the under side of the bed is an arc of 5327 feet length. The difference of 190 feet in the lengths of the two arcs gives at least some measure of the rock-flow which must have occurred in the bed during its period of elevation.

But what one sees in the Wills creek anticline locally and partially, and understands only in a dim and nebulous way, has been studied over wide areas, and correlated and explained by those fully equipped for the work. The bended strata of a part of western Europe have been fully illustrated by Heim.* And though Reyer† has unfavorably criticized part of his work, it stands yet good so far as concerns our present purpose. The United States Geological Survey has published two monographs of exceeding interest to readers of the literature of mountain

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* *Mechanismus der Gebirgsbildung.* † *Theoretische Geologie.*

structure: that of G. K. Gilbert on the "Geology of the Henry Mountains," and one of Bailey Willis, entitled "The Mechanics of Appalachian Structure."

So far as concerns the bending of rock-strata, the work of Mr. Willis is not equalled by any other to be found in all the literature of mountain-building. Plate XLIX. of his monograph shows the bent Silurian strata which form the striking anticline to be seen near Hancock in Maryland. In the legend to the following plate, he calls this form a "symmetrical or upright fold, open." The next figure of the plate shows this fold when it is geologically older, and when the left side of it has become steeper than the right, forming what he calls an "inclined fold." In Fig. 3, the flanks of the fold have grown still steeper; and in the syncline to the left of it, one of the strata is shown bent upon itself, like the letter V, and in its lower part welded together as the two sides of that letter are welded at its apex. The fold has "closed." Fig. 4 shows the fold forced still further over towards the left, until it has become "closed and overturned." In that axis of the fold where exists the greatest pressure, where it will shear off if further pressed, giving birth to a fault, there the more yielding strata have become greatly attenuated. So much of their mass as has been forced away, has flowed into the adjacent parts of the fold, where evidently the pressure is least.

Plate LI. shows two figures which illustrate what may occur in a "squeezed syncline." The left figure might be simulated by a sack much constricted at two-thirds its height, and filled to overflowing. In the right figure, the stricture is entirely accomplished; the contained material is divided into two portions. In effect, here is illustrated a rock-mass contained in a harder rock-sack which almost is closed; and finally is closed entirely.

Although our author meant, in these figures, to illustrate only the folding of rock-strata, yet he has taught us clearly that rock-masses, when under pressure, flow freely. And that he does so maintain, he stated to the writer in a recent interview.

That the flow of rocks has deformed objects contained in them, foreign to their magma, is a fact broadly known. It was through the study of elongated and compressed fossils that there came to us our first correct conceptions of the "cleavage"

in slates. In one of the discussions to which Tyndall's active mind often impelled him, he treated of the subject in a Royal Institution lecture, given in 1856.* Heim, already cited, gives figures of similarly distorted fossils; one of a belemnite pulled asunder at many points, having the interspaces filled with calcite, while the rock upon which the fossil rests has flowed without breakage.

In the building of the Geological Survey at Washington, there are to be seen quartz pebbles, or small quartz bowlders, which have been drawn out to perhaps double their normal lengths. That these are not illustrations of aquo-igneous fusion is shown by the schistose magma in which they are in part imbedded. And that these examples are not isolated and peculiar, that they are indeed abundant, is made evident by George F. Becker, in his paper relating to "Finite Homogeneous Strain, Flow and Rupture of Rocks,"† in which may be found this passage:

> "Yet there is no doubt among geologists that pebbles, even of quartzite, in conglomerates are not infrequently elongated by pressure to double their original lengths without rupture."

The flow of rocks is a part of the general subject of the plasticity of solids, and it has a close relation to the flow of metals, concerning which it is quite unnecessary for me to present any observations to the Institute, a body comprising those who know most about that phenomenon, and whose daily occupations require that they shall diligently keep its conditions in mind.

Many earnest workers have studied the movements of rock-strata reaching over wide areas; their bending and breaking, elongation and shortening, their comparative rigidity and plasticity. Considerable work has been done in the field of the aquo-igneous fusion of rocks, a condition which adds greatly to their plastic movements; and as was to be expected, our industrious brother, Sterry Hunt, has left us much of it in his writings. But I have not been able to find that any one has taken for a theme those entirely local movements which cause country rocks to encroach upon mine-openings; movements which I

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* *Fragments of Science*, New York, 1871.

† *Bull. Geol. Soc. Am.*, iv. 13.

have ventured to attribute to the flow of rocks. It is true that they may be referred to those forces which cause the far extended foldings of rock-strata; the stress which is supposed to grow out of shrinkage in the earth's outer crust. This cause may be supposed with all the more justice if the mine be in a disturbed region. But in the undisturbed Lower Carboniferous horizon of the Acadian coal-field, as at Albert, and among the Upper Carboniferous rocks of the West Virginia coal-basin, as at Ritchie, it would seem more philosophical to refer to rock-flow the closing-in of mine-walls.

As has been mentioned already, Ritchie lies in a gentle synclinorium, which measures more than 150 miles across. The rocks there rest in undisturbed, horizontal strata. Moreover, the axis of the fissure is nearly vertical to the axis of the syncline; it stretches across the trough of the wave and in the center of it. The stress which gave birth to the syncline, if still active, would tend to shorten the cleft and not to move its two walls nearer each other. These conditions, and the inference, should be kept in view, more especially because a learned and honored student has told us that the Albert vein-walls tended to approach each other because of the "crowding of the Hillsborough rocks toward the northwest." Such a condition is not true at Ritchie; yet the two cases are similar.

The writer begs permission to offer the subjoined hypotheses. If they have little weight, they necessarily must have some value, even if negative. They will prove at least something to object to; and it will be by elimination of error that finally we shall reach the truth.

1. Following the usual order of occurrences within the earth's crust, the Albert and Ritchie veins were results of slowly augmented strains, which at last produced rupture.

2. The resulting fissures slowly increased in width and length until the rupturing stress no longer existed in the rocks. From that time to the present the fissures have been growing narrower, because of the flow of rocks when under pressure.

3. The flow has been least in the obdurate strata and greatest in the most yielding ones. It has been slowest in the heavy bed of sandstone at Ritchie, and most rapid in the soft and greasy black shale in which Albert lies.

4. The rupturing force shattered the rocks along and near

the potential surfaces of rupture, thus generating in them many small surfaces of decreased adhesion, which afterwards were developed fully, permitting the detached masses to fall away into the fissures.

5. Where the fissure-walls were sandstone, these detached masses were comparatively broad and thin; they were the sandstone flakes found while removing the Ritchie vein-matter. Where the fissure-walls were shale, the detached masses had no definite figure, and decidedly were not flakes or slabs.

6. Because of the homogeneous quality and rigid character of sandstone, such strata pulled asunder promptly; there was but little shattering along the coming surface of fracture. Because of the yielding and readily-flowing qualities of shale, there were engendered many planes of slight adhesion. The shattering or collateral sub-fissuring was longer continued and more profound.

Hence, the sandstone-walls at Ritchie were smooth and nearly coincident, because the detached masses which had fallen away from them were few, and each was thin and of large area.

Because of profound shattering at Albert, very considerable quantities of stone had fallen from its walls, leaving a vein extremely irregular in width, but always narrowest and most uniform where the strata were hardest (and therefore least plastic and least crushed), as Prof. Hitchcock has pointed out.

7. The shale strata at Ritchie were protected by the sandstones above and below each of them. Also, they were more rigid than the Albert shale. Hence, they were less crushed by the rupturing stress; their walls were less irregular.

As soon as the stresses were neutralized in the Ritchie shales, they at once began to flow inwards upon the empty fissure. They flowed faster than did the sandstones. Hence, when grahamite entered the cleft, the vein between the shale-walls was narrower than it was in the sandstones.

8. The pressure which observers declare to have existed upon the vein-matter, both at Albert and at Ritchie, grew out of this flow of the rocks.

9. From the formation of the Ritchie vein to the injection of its enclosed matter, the time which elapsed was considerable, and greater than was the corresponding period at Albert.

Notes on Certain Water-Worn Vein-Specimens.

BY F. C. HOLMAN, SAN FRANCISCO, CAL.

(Atlanta Meeting, October, 1895.)

It is desired in these notes to record a vein-phenomenon certainly unique in the writer's limited experience, and, as it seems to him, sufficiently rare to be worthy of mention.

In the little mining camp of Forbestown, situated in the Sierra Nevada of California, it was announced one day, when the miners of the Gold Banks mine came off shift, that "washed gravel" had been struck on the 265-foot level. To those acquainted with that section and the character of the mine (a gold-bearing quartz-vein) this was a somewhat startling announcement; for, prolific as the State is, we are not in the habit of mixing up promiscuously our vein- and placer-deposits. However, the statement proved to have some foundation.

The mine is located at an altitude of about 2700 feet, on the brink of one of the great canyons of erosion characteristic of the drainage of this range. A lateral ravine cuts the vein at right angles, thus making its general strike approximately parallel to the canyon at this point, and with a variable dip of from 20 to 40 degrees towards the canyon. The strong, well-defined vein, often 20 feet wide and more, presents nothing to distinguish it particularly from scores of others in the State, unless it be considered peculiar in the highly crystalline character of its quartz and the nature of the enclosing country-rock, which is metamorphic in the extreme, very hard, devoid of lamination, and in bedding so obscure that it might easily pass—at least on superficial examination—for one of the basic eruptives. Near by in the district can be found a series of slates common to the gold-belt of this range, and within the limit of the company's property is an intrusion of an acid eruptive. On an adjoining claim a granite shows itself. The mine is opened through an inclined shaft sunk on the vein. The ore carries both free gold and sulphurets, the greater part of its value

being in the latter. For a California gold-ore, the amount of silver is unusually high. The vein being very compact and the quartz highly crystalline, the mining of the ore is by no means easy. Except at the immediate surface, no oxidized ore is found. Vugs are numerous, furnishing large and perfect crystals, and it is with one of these cavities that we have to deal.

It was opened into on the roof and side of the 265-foot level; irregular, chimney-like and of such size—running upwards on the dip of the vein for a rod or more—as to permit one easily to crawl into it. It gradually closed in at both ends. Unfortunately, the work going on in this part of the mine made it impracticable for a visitor to trace the continuation of this cavity (if any existed) in either direction, or to form a more thorough idea of its immediate surroundings. The foreman said it could be traced to the level above, 100 feet.

As far as it was possible to judge, the vein at this point was in its original condition, with solid quartz on all sides of the vug, and no evidence that the vein had been fissured or broken up by any movement of the walls since the deposition of the quartz. There was no water coming in on this level and very little on the level above.

The sides and roof of this cavity were beautifully studded with crystals, many of them large and perfect, while upon the inclined bottom was a mass of loose quartz-débris. Generally scattered over the bottom, the deposit increased in quantity towards the lower end. It comprised both amorphous and crystalline material, single broken crystals and clusters. While most of this material was rather fine, the majority of the pieces being under 3 inches on any dimension, coarser stuff was mixed with it, some of the latter masses, which had evidently fallen from the roof or sides, being slab-like in their proportions and suggestive of a crustified deposit in process of formation.

While the loose stuff furnished plenty of pieces not defaced further than by simple fracture, a large proportion of it showed evidences of attrition in all degrees, from the crystal with only the apex and edges slightly smoothed off to the rounded, translucent pebble, such as any water-course or sea-beach might produce. Admitting a preliminary fracturing, a close exami-

nation of the specimens leaves little room to believe that any other agency than that of mechanical abrasion in a stream of water will account for their later reduced condition. The wearing-down of the most obtrusive points and the corresponding protection which others, less exposed, have received; the irregular effect in the same specimen of a rough grinding here and a polishing there; the accumulation of fine quartz particles and dust in the cavities among the crystals; these alone, without other considerations, would seem to establish sufficiently the fact of this agency.

There has been a current of water during the formation of this ore-body that in its velocity departed very materially from the slow percolation usually associated with the genesis of ore-deposits of this kind. To which of several causes the fracturing is to be ascribed is not so evident. It looks as if the violent action of the water might be responsible for some of it. If not wholly due to this cause, the balance must be charged to the account of a movement in the walls of the fissure, or to gravity acting in the simple form of masses falling from the roof, a few feet to the floor, but not through the buoyant medium of water. Besides the crystals that have been both broken and eroded, many showed no signs of the latter effect, and while some of these might have been recently broken in mining, the evidence of fracturing by natural causes, is both too apparent and abundant to allow this possible human agency to have any bearing of importance. One, at least, of the main factors in reducing some of these masses to their present size must have been coarse material rubbing upon itself; and the force of the stream that could perform the work of moving some of the larger pieces, the entire superficial areas of which have suffered by erosion, must have been considerable. Nor could the appearance of these specimens be the result of a long-continued impact of fine material, acting on the principle of the sand-blast.

Take for example a translucent specimen weighing 19½ ounces avoirdupois, of an approximate elliptical section, 5½ inches long, 3 inches wide and 1½ inches at its greatest thickness. From its original form it must have lost, at a low estimate, at least its present weight. Its present rather smooth surface has outlined upon it the bases of numerous large crys-

tals, irregular hexagonal polygons and other cross-sections depending upon the positions occupied by the crystals. While smooth, it is not polished, excepting an occasional small facet-like face where a body in contact by long-continued action has reduced the rougher surface. Such little areas are distinguishable from unaltered portions of crystal surfaces. Most of the crystals have been left with a rough surface; but in others polishing has followed, producing the appearance which a clear crystal has, when its edges have been rounded off by solution.

Iron and copper pyrites in small crystals are found in the specimens, and partake of the same general effects as their harder surrounding matrix.

Equally interesting with the fact of attrition is that of deposition as displayed in a very thin coating of iron sulphide upon the eroded surfaces. As a coating, it is too minutely crystallized to allow its form to be made out under an ordinary glass; but it has in appearance nothing in common with the older and coarser deposition of iron pyrites, contemporaneous with the quartz. Very firmly attached, it is present on all of the larger specimens in patches, and must have been widely deposited in the chamber upon the *débris;* and its absence on parts of these same pieces can clearly be ascribed to erosion, all the phases of which are as plainly shown on this metallic coating as upon the underlying eroded quartz surface.

Here, then, in a large fissure-vein, along one of the closing channels of its formation through which circulated waters carrying the vein-forming material—quartz and its accompanying sulphides—mechanical work has been performed by a current under sufficient intensity of pressure to grind quartz upon itself. The solid worn masses and the diminution in their size, and in the size of large crystals (particularly such as are imbedded and exposed along the plane of the major axis, so that they could have lost but little if any by fracturing) tend, with other details, to show a prolonged action or a violent one, or some combination of the two. It is hard to conceive of a current born of the deep circulation as responsible for this case. Yet if the erosion is a phenomenon of the vadose circulation, as seems probable, so also is the secondary deposit of sulphide, and we are forced to accept somewhat peculiar attending conditions to account for it. It would be a case of mineralization by the

two circulations at the same point, a considerable distance below the surface; the metallic sulphides being drawn from two distinct sources of supply, far removed from each other. Except in limestone regions and certain other formations, a stream pouring directly into a fissure depositing and eroding alternately is not commonly displayed in the shallow circulation. Furthermore, the erosion was not chemical but mechanical.

Other accessories to the fact must be allowed: an oscillation in the stream to permit two such antagonistic forces to act; a sufficient supply of organic matter to reduce the oxidized products of other veins, carried in solution; an open channel and free exit somewhere for such a volume of water. The present topography of the country on the strike of the vein does not suggest that any such opening, offering the required difference in level, existed; and it would be necessary to look for an outlet in a cross fissure at right angles leading into the present drainage.

Such an assumption will of course require, for the age of this erosion and secondary deposition of mineral, a very late geological date, subsequent to the cutting down of the country and the formation of the canyon and its drainage-system.

The Eastern Coal-Regions of Kentucky.

BY GRAHAM MACFARLANE, LOUISVILLE, KY.

(Atlanta Meeting, October, 1895.)

THE eastern coal-field of Kentucky covers about 11,000 square miles of territory. On the east and south it extends to the State lines of West Virginia, Virginia and Tennessee. Its western boundary may be roughly described as an irregular line beginning at the Ohio river, in Greenup county, thence through the counties of Carter, Menifee, Wolfe, Estill, Jackson, Rockcastle, Pulaski and Wayne, to the Tennessee line.

The changes in the rock-intervals and in the coal-seams from the northeastern end of the field to the south and southeastern end are so great that no satisfactory identification of the coals has yet been made, and the correlation of the Kentucky coals

with those of other States in the Appalachian coal-field is likewise involved in a conflict of opinion. The longer the matter is studied, the more confusing it becomes. The first examination of this field leads to much more satisfactory conclusions than can be accepted after more thorough investigation; and the identification of the various coal-seams, supposed to be established forty years ago,* is now doubtful.

This paper will, therefore, be confined to the description of the various mining operations, with some information as to the territory adjacent to, or tributary to, developed sections, using generally local or Kentucky nomenclature; and it will give also some observations on the stratigraphy and correlation of the coal-seams, which may be useful in stimulating study by geologists.

Kentucky, in common with the rest of the South, must suffer severely from the fact that much of the development, and most of the investment of capital, was made during the boom. Ten thousand acres of land, to which the boomer either had a defective title or no title at all, was all the foundation required for a coal and iron company. By fortunate chance, a few investors got really good property for their money; but, as a rule, the lands proved valueless. At that time the field possessed little interest for the practical engineer. A careful, conservative opinion was the last thing the boomer desired.

This fever having passed over, it is to be hoped that better times are in store for the district, and that really meritorious properties will receive the attention which they deserve. The difficulties about the titles are not insurmountable, and the mountain-feuds need not interfere with mining operations, as there is no necessity for strangers to take a hand in them.

The northeastern mining district comprises Carter, Boyd, Lawrence and Johnson counties. The general section, Fig. 1, applies to Boyd and Carter. Farther south the intervals widen, and possibly the seams with the same numbers may not be the same seams.

This district adjoins the Hanging Rock region of Ohio, and the sections should be quite similar.

The Kentucky scheme of correlation is as follows:

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* Kentucky First Geological Survey.

Kentucky.	Ohio.	Pennsylvania.
Coal No. 1,	Jackson Shaft,	Sharon.
Coal No. 3,	No. 3,	Lower or Upper Mercer.
Elkhorn, Jellico, Peach Orchard,		
Coal 4,	No. 4.	
Coal 5,	No. 6.	Brookville.
Coal 7, Ashland,	Sheridan. Hocking No. 10.	
Coal 8,	No. 11.	Lower Kittanning.

In Carter county, coals 3, 4 and 7 are being mined to some extent. No. 3 is generally less than 3 feet, including a small bone parting, the output being mainly sold for steam. No very large or very profitable operations can be based on this coal. At times this seam may become cannel. The cannel coal-mines of the Kentucky Cannel Co. are either in No. 3 or a lower split of No. 4.

Coal 4, the "Hunnewell" cannel-seam, is the one from which the rich gas-enriching cannel of the Lexington Mining Co. is obtained. It is extremely sporadic, and probably operations thereon will prove short lived. About 5 miles south of Willard is a pocket of cannel which looks very promising. The seam averages: bituminous, 12 inches; cannel, 18 inches. This seam shows cannel in many localities in Carter and adjoining counties, and as the quality is generally good, will justify examination and possibly investment. The mines which work coal 7 in Carter county are generally nearly exhausted, but there is considerable territory yet undeveloped south of the Chesapeake and Ohio Railroad toward Willard. The coal-production of Carter county in 1893 was 132,000 tons.

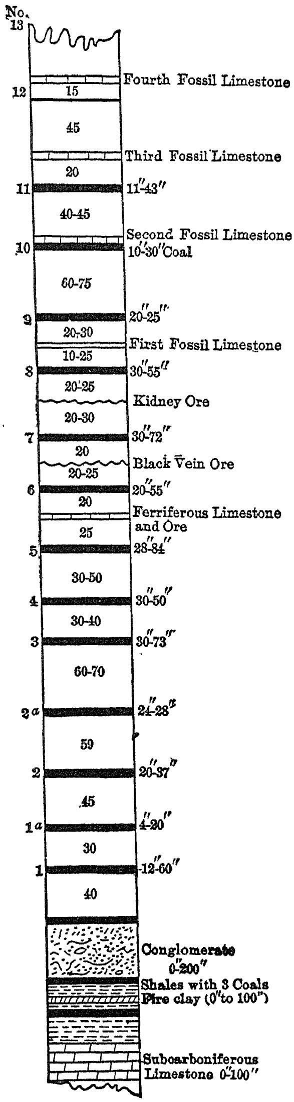
In Boyd county are the extensive mines of the Ashland Coal and Iron Railroad Company, working seam No. 7. The average bed-section is

	Inches.
Slaty top-coal,	6 to 24
Coal,	24
Slate,	1
Coal,	24

The coal produced is a very fair quality of domestic and steam-coal, and well adapted for use in the blast-furnace, where,

of late years, however, it is used in connection with Kanawha or New River coke. The remainder of the output is sold to

FIG. 1.



Section in the Northeastern District of Kentucky. The figures printed in the intervals indicate feet of vertical thickness. The thickness of each coal-seam is given in inches on the right.

steamboats on the Ohio or is shipped by rail to central Kentucky. Boyd county produced 153,000 tons of coal in 1893.

Going from Catlettsburg up the Big Sandy valley to Peach Orchard, Lawrence county, we find the mines of the Peach Orchard Coal Company, said to be working seam No. 3. If so, it has much enlarged and become badly split up. However, by dint of much care and labor, an excellent coal is produced from a very dirty bed. Lawrence county produced 94,000 tons of coal in 1894.

On Nat's creek, near the top of the dividing ridge, is found a very large seam of coal, very high in the Measures. The lower part of this seam is a dense block-coal, which might answer for the blast-furnaces at Ashland. This seam is interesting as probably belonging to the Lower Barren Measures, and apparently corresponding to the Big Bed at North Coalburg in the Kanawha valley, West Virginia, which is believed to be the Mahoning coal of Pennsylvania.

Eight miles above Peach Orchard at Whitehouse, in Johnson county, are the Chattaroi and Birdseye cannel coal-mines, working in seam 2 (?) which in this neighborhood is quite generally a cannel-coal and produces fuel peculiarly adapted to open fireplaces. These mines, representing modest investments, have been very successful. The cannel is about 2 feet thick, capped by a mining-bench of bituminous coal. A mile above Whitehouse, coal is mined in the bed of the river said to be No. 1, which at Paintsville, 15 miles up, is 150 feet above the river and is upwards of 4 feet thick. On the west side of the river, opposite Paintsville, on Miller and Daniel's creek, is a good field of this coal in good condition for mining, and running 5 feet, mainly splint- or block-coal. Just above Paintsville, seam No. 4 (?) has a big exposure of good cannel-coal known as "East Point" cannel. This extraordinary thickness of 4½ feet I believe to be strictly local; and, furthermore, this locality being up on the Paintsville anticlinal, there are probably no large areas of this seam, the cannel, more likely, lying in isolated patches. This deposit has caused a great deal of wild talk, and the exaggerated statement that there are 5000 acres of cannel-coal 5 feet thick, has been often heard.

Going up the river, there is nothing of special interest till we reach the upper part of Floyd county and Pike and Letcher counties, which is called the Elkhorn coking-coal region. Doubtless the eastern part of Pike carries the "Thacker" coal

of West Virginia, now mined on the Norfolk and Western railroad, and also the Elkhorn coal.

Remembering that the measures thicken along our course, it is not surprising to find the section greatly enlarged, as shown in the general section by Mr. Crandall, given in Fig. 2.

The Elkhorn coal, called No. 3 on this section, reaches in places a thickness of 9 feet and generally, though not universally, has a structure indicating a coking-coal, which experiments in small ways confirm. The seam is not a very persistent one,—but large areas of good thickness are pretty safely defined, and the region has a great prospective value. Some of the other seams look very promising, and with further development may add greatly to the resources of this region. The "Flat Woods coal" is of interest as being the highest we have so far met, and possibly corresponding to the Nat's creek coal (Big Bed of Kanawha).

What is supposed to be this same Elkhorn seam No. 3, extends (after being cut out by the Pine mountain fault) into Harlan county, and so on to the Norton and Big Stone region. Similar conditions, possibly on a reduced scale, may be looked for on the projected extension of the Lexington and Eastern (formerly Kentucky Union) Railroad.

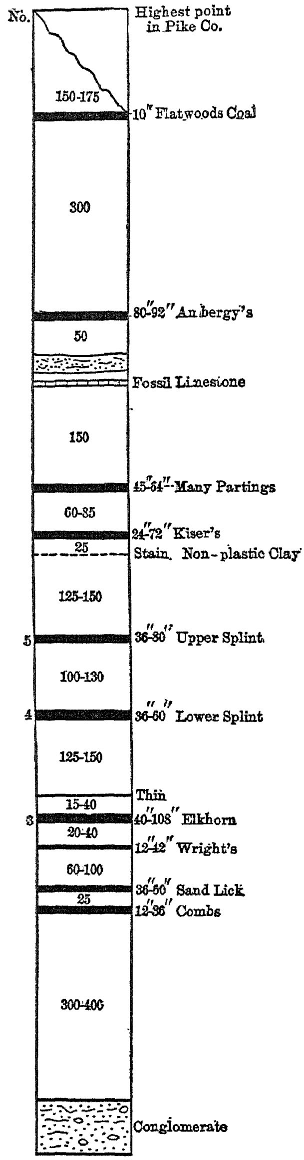
In Lee county, on a branch of the Lexington and Eastern Railroad, near the rim of the basin, there are three mines producing the Beattyville coal from what is conceded to be a Conglomerate (XII.) coal-seam. The coal is less than 3 feet thick, rather limited in area, uncertain in its occurrence and expensive to mine; but it is a most excellent coal. On account of the high cost of production, it will never hold a prominent place in the market, but will always (by virtue of its location and quality) be in more or less demand in Central Kentucky. This place was once the scene of an active boom.

In Breathitt county, beyond the terminus of the L. & E. R. R., and also in adjoining counties, are several areas of cannel-coal which will be valuable. The generally accepted views of this region are greatly exaggerated; but with the extension of this railway some good territory will be opened up, including the cannel-fields of George's branch and Troublesome creek, and also in Morgan county on Caney creek.

Going now into the southeastern mining district, and passing

some abandoned coal-mines in the Conglomerate (XII.), we come

FIG. 2.



Section in the Elkhorn District. The figures printed in the intervals indicate feet of vertical thickness. The thickness of each coal-seam is given in inches on the right.

to the Laurel coal-field. The coal mined here, called No. 1 by the State Survey, is always under 4 feet, frequently as low as 3,

and is a fair grade of coal for steam and domestic purposes. It is not as good as the Jellico coal, which is mined from a higher seam. Laurel county produced 286,000 tons of coal in 1893. The Laurel coal is 50 to 100 feet above what is assumed to be the top of the Conglomerate, which latter contains, just west of here, three seams of coal in Rockcastle, Wayne and Pulaski counties, the Bryvan, Main Cumberland and Barren Fork coals; the latter about 200 feet below the top of the Conglomerate. This region is particularly interesting as showing the western outcrop of the Conglomerate Measures, which here carry three coals or groups of coals as shown in the general section, Fig. 3.

The lower part of this section also suggests correlation (if that be possible) of the coals of the Eastern with those of the Western (Illinois) coal-field of Kentucky.*

The northernmost mine on the Jellico seam (No. 3) is in Knox county at North Jellico. The coal is very hard, mined chiefly by machinery, runs nearly 4 feet thick and is mainly marketed in Louisville. Knox county produced 160,000 tons of coal in 1893.

Farther south in Whitley county and beyond the Tennessee line is the Jellico district, where the most extensive mining industry in the State has been developed. The Jellico seam averages less than 4 feet, frequently with a small parting, and produces a most excellent coal, which commands the highest market price wherever it goes. A portion of the mines are on the Tennessee side, but are mainly owned by Kentuckians. East of Jellico is mined what is known as the Bird-Eye coal, a peculiar and unique variety, with a knotty structure similar to birdseye maple or walnut. It is an excellent domestic coal, but the deposit is not likely to be extensive or reliable. About 400 feet above the Jellico is the Vanderpool seam, 3 to 6 feet, producing a hard block-coal of very good quality.

The region west of Jellico, toward the Cincinnati Southern Railroad, has been more or less carefully prospected, but so far with rather disappointing results. The production of Whitley county in 1893 was 456,000 tons.

East of Jellico and on the Cumberland Valley division of the L. & N. Railroad are the Pineville and Middlesborough mines.

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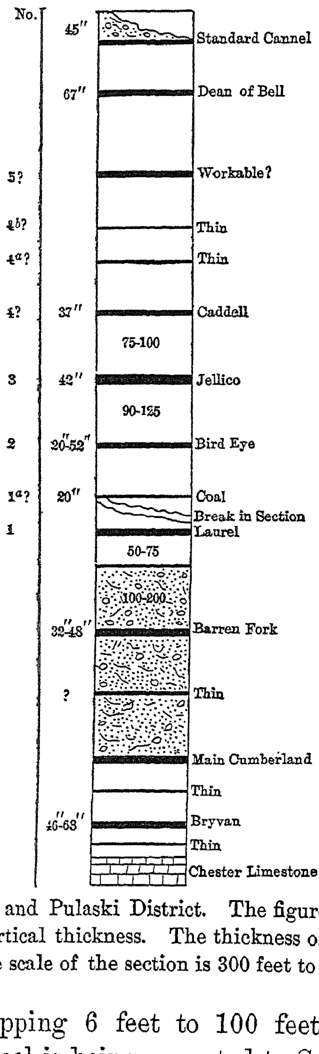
* See section in Ohio county, *Kentucky Reports.*

Twelve miles from Pineville the Bear creek cannel coal-mines are working a remarkably thick deposit of cannel coal, the seam being near the base of the Lower (?) Coal Measures, possibly 300 to 400 feet above the Conglomerate (?). The seam, 4 feet thick, is in the basin between Pine mountain and Cumber-

FIG. 3.



Section in Whitley and Pulaski District. The figures printed in the intervals indicate feet of vertical thickness. The thickness of coal-seams is given in inches on the left. The scale of the section is 300 feet to the inch.

land mountain, dipping 6 feet to 100 feet away from Pine mountain. This coal is being exported to South America and Europe, and is also sold to American gas companies for gas-enriching.

The Pineville Syndicate is mining a seam of cannel-coal about

700 feet above the Cumberland river-bed, its section being bituminous 40 inches, cannel 20 inches. This cannel is sold mainly in South America, the bituminous portion of the seam being sold to the L. &. N. Railroad for fuel.

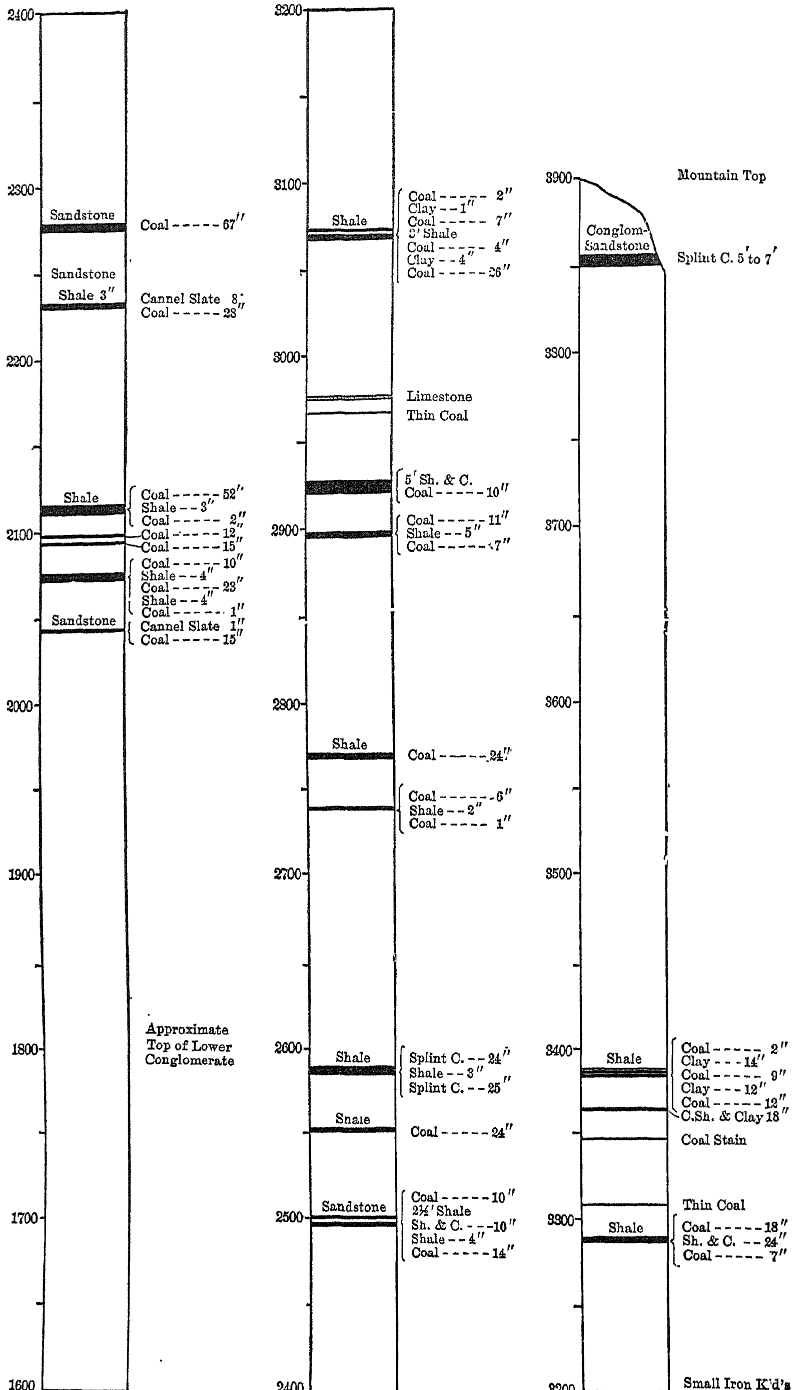
Near the foot of the mountain, being placed as No. 3 (Elkhorn), is a little seam of coal about 30 inches thick. Several mines have been opened and 250 coke-ovens built. The coal is very pure, high in volatile matter, and makes a pure coke, but one of very weak physical structure. This seam of coal, coupled with a little bunch of brown iron-ore, found on the outer edge of Pine mountain, was the basis of the late lamented Pineville boom. A very large amount of money has been sunk here, not to mention the losses in town-lots in Pineville.

We continue, what to many will be a painful journey, to Middlesborough, the "Magic City" whose mushroom growth and wild speculation are almost without a parallel in American history. Hotels here and at Harrogate were erected at a cost of hundreds of thousand of dollars; millions were invested in a steel-plant and blast-furnaces; town-lots, intrinsically worth less than fifty dollars per acre, sold for three hundred and fifty dollars per front foot; and mountains were cut down with steam-shovels to fill holes and level the land for building-purposes. The *raison d'être* of this boom was two or three thin seams of (Clinton) iron-ore on the south side of the Cumberland mountain; some pockets of brown ore near by in Tennessee; and, on the Kentucky side, a fair quality of coal, out of which, after years of trial and failure, a tolerable coke is now made, and a very fair quality of steam and domestic coal is mined. There is some justification for the development of the coal-mines; but the iron deposits have proved of little value. The coal-mines are on Bennet's fork of Yellow creek, the seam being upwards of 5 feet thick, and cheaply mined. This enterprise will probably prove useful, and reasonably profitable.

The section in this region is greatly expanded. Which seam is being worked is not known; but it is apparently pretty high in the Measures.

Going up the Cumberland river, we come to the Big Black mountain, which extends into Wise county, Virginia. Conditions on the Kentucky side are quite similar to those in Wise county which are described by Mr. Hodge, in his paper on

FIG. 4.



SECTION FROM LITTLE BLACK MOUNTAIN.

From the paper of Mr. Hodge, *Trans.*, xxi., 925.

the "Big Stone Gap Coal-Field,"* and evidence a very promising coal-field. Figs. 4 and 5 are sections reproduced from Mr. Hodge's paper.

The identification of the seams in the various sections in Kentucky, and correlation of the Kentucky seams with those of other states, seems now to be involved in obscurity and confusion. Aud, furthermore, the difficulty of tracing the various members of these series is increased by crossing the Pine mountain fault, as well as some other local irregularities.

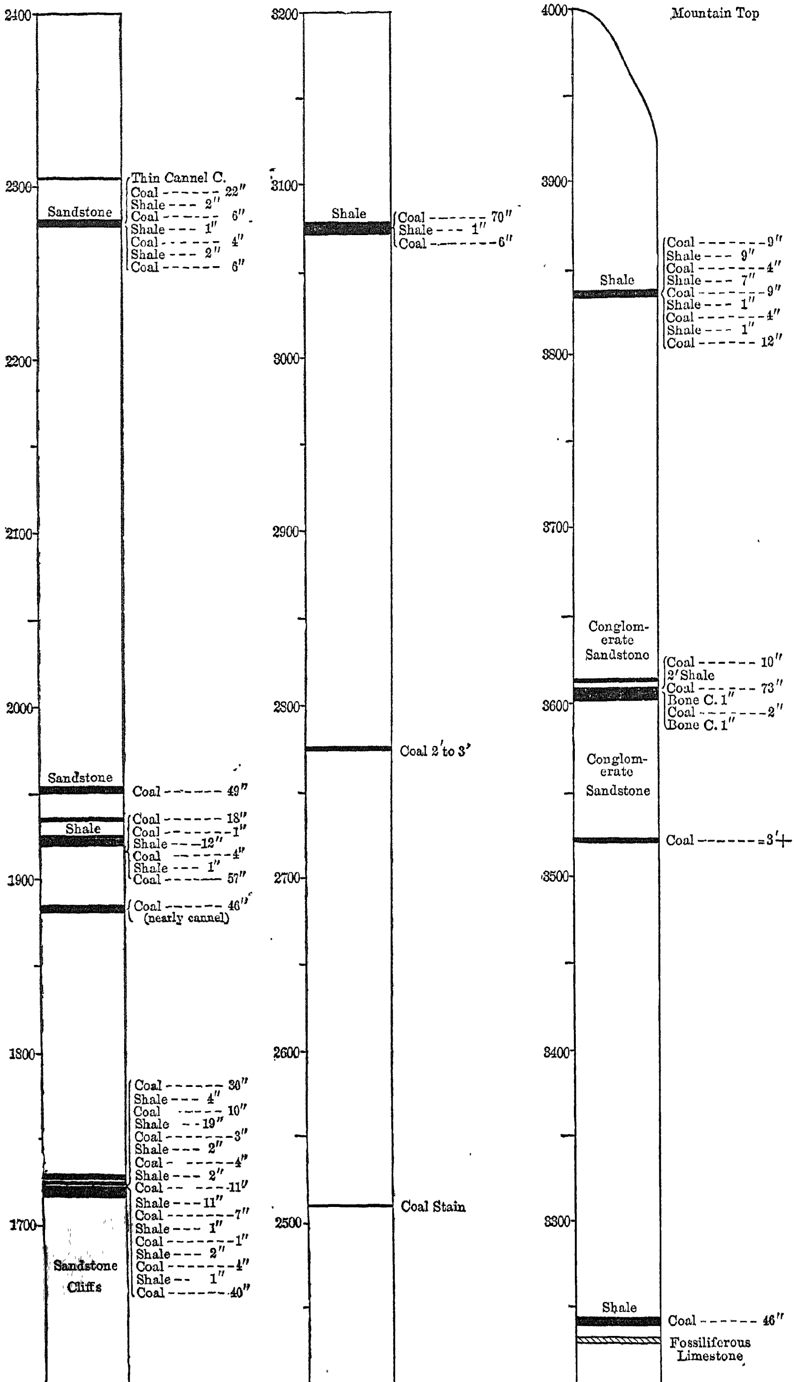
The Kentucky geologists correlate with Ohio, thereby entering the State in a district where the section is very much shortened. Seam No. 1 in the Boyd and Carter section (northeastern Kentucky) is considered the Jackson shaft = Sharon (Pennsylvania). The Peach Orchard coal of Lawrence county is called No. 3 by the Kentucky Survey = the Upper or Lower Mercer of Pennsylvania; whereas the work of I. C. White, which I think will be confirmed in the mind of any engineer who will go over the ground, places Peach Orchard pretty high in the Kanawha or Lower Productive Coal Measures. Again, the Kentucky Survey identifies the Laurel as No. 1 = Sharon, whereas the whole of the Rockcastle group is undoubtedly below the Laurel.

Possibly a broader view of the situation might be had by entering the State from a region where the whole of the Coal Measures from Pottsville No. XII. to the Pittsburgh Coal Measures inclusive is displayed on a grand scale, with the lines of demarcation and the ear-marks of each division so well marked that, at the start at least, there is very little chance for error. On New and Kanawha rivers, in West Virginia, each series, the Pottsville XII., the Lower Coal Measures, the Lower Barrens, and the Pittsburgh Coal Measures are so clearly defined that their location is evident to most casual observers. Furthermore, several well-records aid in tracing the rapidity of the thickening of the Measures.

In the New River coal-region, Pottsville XII. is 1400 feet thick, embracing three coals or groups of coals, as compared with 600 feet at Charleston and less than 300 feet in Ohio. Going down-stream to where the top sand-rock of XII. goes

* *Trans.*, xxi., 922.

Fig. 5.



SECTION FROM BIG BLACK MOUNTAIN.

From the paper of Mr. Hodge, *Trans.*, xxi., 927.

under stream-level, we find the Lower Coal Measures 1000 feet thick (capped by what appears to be the Mahoning member of the Lower Barrens), as compared with 600 feet near Charleston and 300 feet in Ohio. Near Charleston the Lower Barrens are 800 feet thick, perhaps 200 feet thicker than further north. Farther down-stream the Pittsburgh Coal Measures are plainly visible.

The characteristics of each member of the series are almost typical; the variations are mainly in the thickness. As to the XII. (Pottsville) and the Lower Coal Measures, practically the same status occurs on the head-waters of the Guyandotte and Big Sandy rivers, near the Kentucky line. The coming-up of the Pine mountain fault, and the difficulties of identification along the Clinch and head of Big Sandy in Virginia, interpose considerable difficulty in the transit into Kentucky.

To account for the great thickness of the section in Wise county, Virginia, and Harlan county, Kentucky, it seems to be the impression that we must call in everything from the Pottsville to the Permi-Carboniferous. But so far as my observation goes, I do not believe that any part of this section, or any other in Kentucky, reaches higher than the Lower Barrens.

So far as known, the coal-seams of No. XII. are rim-deposits. Borings have rarely located them very far into the basin. But along the southeastern edge of the basin the Conglomerate Measures attain a magnificent development. The Lower Coal Measures also show a great thickening as we near the southeastern part of the field.

Mr. Boyd says that he has traced the Conglomerate above the Pocahontas coal across Virginia, and identified it with the sand-rock above the Banner and Imboden coals of the Clinch valley. These coals have been *assumed* to correspond to the Imboden group shown in the lower part of Mr. Hodge's section; and the Kentucky Survey locates the Elkhorn or Jellico at the Imboden. This involves us in a mass of inconsistency, since far below the Elkhorn or Jellico seam we have the Laurel (Sharon) coal, and below this the whole of the Rockcastle group, which would involve the admission of an entirely new series of coals below the Sharon. This we are hardly prepared to admit, although in the northeastern district there are some intra-Conglomerate coals of no commercial import-

ance, which are apparently below coal No. 1 (Sharon), and which may possibly expand into the Rockcastle series. There is serious doubt as to the Clinch Valley Imboden coal being equivalent to the Imboden; and I consider it questionable whether the Elkhorn is equivalent to the Imboden. I cannot reconcile myself to the belief that the Laurel coal is equivalent to No. 1 of northeastern Kentucky, which in turn is equivalent to the Sharon. I would read the Elkhorn creek section as embracing coals from the *top* of XII. to and including the Mahoning coals of the Lower Barrens. I think the Laurel coal is in the Lower Coal Measures, and believe the Rockcastle group covers the whole of No. XII., the Conglomerate.

In the Wise county, Virginia, and Harlan county, Kentucky, section, the coals in the Double Tunnel at Big Stone Gap are undoubtedly in XII. Whether the top of XII. is above the Imboden, etc., group has not been determined; but I do not think it is so.

It is, of course, possible that the coals in the upper part of XII., which in Pocahontas do not cut much figure, may here, in the profusion of nature, expand into important coal-seams. The Pottsville is generally tripartite, and, but for the tact that Mr. Boyd claims to have identified the Conglomerate rock above the Banner coals of Clinch valley with the Conglomerate rock above the Pocahontas seam, I should incline to place the Banner coals higher in the Measures, and to correlate the Double Tunnel coal with the Pocahontas.

It is possible that the fossil-botanist may be able to identify the upper part of the Wise county section with the Pittsburgh Measures and the Permi-Carboniferous, but there does not now appear to be any other evidence thereof. On the contrary, the evidence is mainly in favor of the theory of great enlargement of the Conglomerate and Lower Coal Measures. As to the attempt to carry the Conglomerate coals in a continuous sheet across the Kentucky field, as our Kentucky authorities do, my observations are opposed to it.

The Magnetic Separation of Iron-Ore.

BY CLINTON M. BALL, M.E., TROY, N. Y.

(Atlanta Meeting, October, 1895.)

MAGNETIC iron-ore is found in many localities throughout this and other countries, in large bodies and in convenient proximity to other materials required for its conversion into iron and steel; and these bodies of ore would have great commercial value on account of their nearness to consuming markets if they were generally suitable in their natural state for use in the larger operations of steel-making.

In the manufacture of iron and steel it has been found, however, that not all ores are equally valuable as sources of raw material for the finished products of furnace, forge and foundry; and, as is now well understood, this is because of the intrusion, in the process of formation of the ore-bodies, of substances which can enter into deleterious combinations with the iron itself, or form compounds associated with the iron oxide which will hinder, if they do not altogether prevent, the production therefrom of the higher grades of finished metal.

It is only since the development of the modern method of steel-making by the Bessemer process that the art of the chemist has been called to a critical study of the nature and effect produced by the presence of these pernicious substances in the furnace while the iron is undergoing reduction to the metallic state. Phosphorus and sulphur are the most common and obnoxious elements, and the admixture of one or the other, or both of these, in small percentages and varying proportions, will greatly modify the final results of manufacture. It so happens that generally, although not always, these hurtful elements are found in compounds in the ore which are not magnetic, and this circumstance has led to the expenditure of much ingenuity and money over a long period of time, in attempts to utilize the attractive power of magnets to separate the magnetic oxide of iron out of its crushed ores and from sands, in

which it is found in great abundance in many parts of the world. An inspection of the Patent Office files of the United States and some European countries discloses hundreds of patents granted to ambitious inventors having this object in view.

Owing to rapid expansion, in recent years, in the world's consumption of iron and steel, and especially in consequence of the very general displacement of iron by steel which has followed the great reduction in cost of the latter, the question has been lately forcing itself upon the attention of steel-makers, Whence are supplies of ores of Bessemer grade to be obtained to meet the ever-increasing demands of the trade? The remarkable developments on the ore-ranges of the Lake Superior district have in part made an answer to this question, but not wholly, for even now doubts are being expressed as to the permanent sufficiency of this source of supply for ores of the Bessemer class, and an advance in price is already noted for the season of 1895 over prices current during 1894. These circumstances contribute to the interest felt in any additions to the visible sources of supply of Bessemer ore which may be available to meet future requirements and insure a continuance of the present low prices for the raw material; it being also felt that the future prosperity of the steel-manufacture in this country will depend very largely upon the ability of our manufacturers to meet the world's lowest prices, thereby insuring full control of our domestic market and a fair share in the trade of the non-manufacturing nations.

The more critical study of conditions essential to economical production, which has followed upon the depressed state of the iron- and steel-business of the country during the past few years, has also led to a keen appreciation, among the most advanced furnace-managers, of the great advantages arising from the use of richer ores in the furnace—an increase in the average per cent. of iron contained in the ore charged to the furnace resulting in increased output of iron, a smaller required amount of flux, an improvement in quality of the product, with less consumption of fuel and general reduction of expense per ton of iron produced. These facts have drawn attention to magnetic concentrates and the peculiar advantages which, when properly selected and prepared for the market, such ores possess

over others of high grade. These advantages consist, not alone in a low percentage of mineral impurities, with correspondingly low phosphorus and sulphur and a high percentage of iron, but also in what is of perhaps even more importance in the manufacture of steel, absolute uniformity of grade—the ore from a particular mine need vary, from time to time and over large quantities, no more than a fraction of one per cent. in the iron contents, with hardly appreciable variations in phosphorus and sulphur. As a factor in steel-manufacture, the theoretical and practical value of magnetic concentrates has been absolutely determined and is freely conceded; there remains, therefore, only the question, which includes others relating to locus of supply and method of treatment, Is magnetic concentration commercially feasible under present competitive conditions? It is one of the main purposes of this paper to set forth the considerations which point to an affirmative answer.

Outside the circle of a comparatively limited number of persons and corporations engaged in the mining of iron-ore and in the manufacture of iron and steel, it is not generally known that a technically perfect method of magnetically separating iron-ore exists; that such a system has been subjected, with success, to the tests of actual commercial experience on a large scale, and that several plants capable of contributing a large amount of ore to the supply of eastern furnaces, are now fully equipped and ready for operation with the resumption of activity in the iron business.

The past contains the record of many failures in ore-separation, and, unfortunately for the projectors, in the record of such failures must be counted some of the most ambitious undertakings. It will not be necessary to dwell upon these, but it will be sufficient, for the purposes of this paper, to point out the road to a successful practice, with some brief reference to what has been accomplished in this art.

In attacking the problem of ore-separation from its practical side, sound theory must govern practice; and the first consideration, after the location of a suitable body of ore upon which operations may be conducted, is, or ought to be, the possible efficiency of the method of separation to be employed. In the early days of the art, the experimenter, fascinated by a mysterious phenomenon—the rush of magnetic bodies, such as

pieces of iron or fragments of magnetite, to a powerful magnet—quickly concluded that very powerful magnets, the stronger the better, combined with any primitive means of moving the ore into and the magnetic portions out of the field of the magnets, would serve all needful purposes in such ore-separation. And again, it was hastily assumed that, given a mass of crushed rock or sand, containing magnetite in a divided state, mingled with non-magnetic gangue, if the attractive power of a sufficiently strong magnet should be brought to bear upon a moving stream of the ore, the magnetite might be cleanly divided away from the gangue by a two-part separation, into concentrates, or heads, and tails—the heads consisting of substantially pure magnetic oxide, and the tails of barren gangue. Moreover, it was apparently never imagined, until a part of the history of several hundred failures had been written into the Patent Office records of the United States and Europe, that any virtue might reside in some peculiar arrangement of magnets, whereby the lines of magnetic force would have a particular and rational arrangement with reference to the moving mass of ore; and it constitutes a serious hindrance to the progress of commercial ore-separation that, even now, persons identified with large interests in ore-mining, and the making of pig-iron and steel, are wedded to, or unable to see beyond these fundamental fallacies, and so, from an experience of failure, rush to the conclusion that magnetic separation as a useful art can have no existence.

To ascertain the efficiency of a proposed method of ore-separation, the investigator ought to be provided with representative samples of the crude ore, the concentrate and the tailings, obtained under conditions of actual commercial performance, and the relative percentages of magnetic and non-magnetic components in each should be definitely determined. The efficiency and practical value of any such system may then be determined by an application of the simple equations given below,* together with a proper consideration of the character and condition of the concentrate, the capacity and durability of the separating-machines, the power and space required for

* The first of these is substantially the same as that given by Mr. Birkinbine, *Trans.* xix., 673.

their operation, and the requisite method of preparation of the crude ore for the best performance of the separators:

Equations.

Let a = percentage of magnetite in the concentrate;
Let b = percentage of magnetite in the tailings;
Let c = percentage of magnetite in the crude ore;
Let d = units of crude ore required to make one unit of concentrate;
Let e = units of tailings for one unit of concentrate;
Let f = percentage of the magnetite in the crude ore which is saved in the concentrate;
Let g = percentage of the magnetite in the crude ore which is lost in the tailings;
Then:

$$(1)\ \frac{a-b}{c-b}=d. \qquad (3)\ \frac{100\,a}{cd}=f.$$

$$(2)\ \frac{cd-a}{e}=b \qquad (4)\ \frac{100\,b\,e}{cd}=g.$$

To convert percentage of magnetite into percentage of iron, multiply by 0.7242.

To convert percentage of iron into percentage of magnetite, divide by 0.7242.

It should be noted that, in ore-separation, in order to meet most satisfactorily the conditions of transportation and use in furnaces, the concentrate should have the coarsest possible granulation combined with the highest attainable purity; the percentage of sulphur and phosphorus being also, if possible, brought within Bessemer requirements. Chemically pure magnetic oxide of iron corresponds to the formula Fe_3O_4, in which combination iron represents 72.42 in 100 parts, and oxygen 27.58. The crystals of this oxide obtained from different mines will vary greatly in size, but will present a general uniformity of character and dimensions when taken from a particular ore-body; and the magnetic ores, at a fair estimate of their general average, may be taken to consist of one-half magnetite and one-half associated rock or gangue. In the concentration of such ores, if the percentage of magnetite in the concentrate be raised to 90, with 4.5 per cent. of magnetite left in the tailings (corresponding to 65.178 per cent. of iron in the concentrate,

and 3.26 per cent. of iron in tailings), such a result may be considered satisfactory practice; as this standard in the concentrate will usually, also, bring the phosphorus and sulphur down to desired limits; and an application of the above equations shows the actual efficiency of the operation to reach nearly 96 per cent., as follows:

	Per cent.
Magnetite in crude ore = ...	50
Magnetite in concentrate = ...	90
Magnetite in tailings = ...	4.5

According to equation No. 1, $\frac{90 - 4.5}{50 - 4.5} = 1.879$ crude units to make one concentrate unit; and, by equation No. 3, we have, $\frac{90 \times 100}{1.879 \times 50} = 95.79$, the per cent. of the magnetite in crude saved in concentrate, 4.21 per cent. being lost in tailings.

Equation No. 4 may be used as a check to prove No. 3; and equation No. 2, to check the accuracy of reports of mine- and separator-operation. The following analysis of the operation will be of interest in this connection:

			Per cent.	Units.	
10 units of crude ore, half magnetite, half gangue, =	5.322 units of concentrate [53.22 per cent.] =	Magnetite,	90 =	4.79	
		Gangue,	10 =	.532	5.322
	4.678 units of tails [46.78 per cent.] =	Magnetite,	4.5 =	.21	
		Gangue,	95.5 =	4.468	4.678
		Total, . . .			10.

By this analysis, as from the former equations, it will be seen that 4.79 units of magnetite in the concentrate $\div$ 5 original magnetite units in the crude ore = 95.8 per cent. of the magnetite in crude saved in concentrate, and $0.21 \div 5 = 4.2$ per cent. lost in tails; also that $10 \div 5.322 = 1.879$ crude units per unit of concentrate.

It may seem to some persons that, in the practice of ore-separation, the obviously desirable results indicated in the foregoing analysis of an efficient operation may be readily attainable by the employment of any mechanically well-constructed separator, without special regard to type, provided only that the separator be capable of developing a sufficiently strong

magnetism to prevent the escape of magnetic matter with the tailings. But this is not the case. A little closer study of the requirements will show that these results involve a more complex series of operations than would at first appear, and that their successful and economical attainment calls for the use of a separator of special design, having co-ordinated functions, out of which the desired results are naturally and easily developed. As has been remarked already, the concentrate should be kept as coarse as possible. This is required, not only to satisfy furnace-requirements and the conditions of transportation, but also to lessen the cost of preparation; fine crushing being expensive. For example, a cube which will pass through a $\frac{1}{4}$-inch hole must be reduced to $\frac{1}{64}$ of its original mass to enable it to pass a $\frac{1}{16}$-inch hole; and the additional cost of such reduction would be in about the same, or probably an even higher ratio; hence the difference in the cost of producing these two grades of fineness might mean, in these times of low prices and fierce competition, all the difference between commercial success and failure.

In addition to the foregoing considerations, a sound theory of ore-separation also dictates that, in the early stages of the operation, the crushing should be carried no further than to just reach the average size of the pieces of pure magnetite, and at this stage separation ought to begin; for a mass of ore brought to this condition will contain some pieces of pure magnetite, some of pure gangue, and others consisting in part of magnetite and in part of gangue; the mixed pieces generally comprising about 10 per cent. of the whole mass. Obviously the most desirable disposition of these three orders of material, while in this comparatively coarse condition, requires that they should be assorted into their several grades. The pure magnetite would then require no further treatment and might at once pass out as concentrate for shipment, and the pieces of clear gangue could be simultaneously disposed of as tailings, thus completing the treatment of nine-tenths of the mass. The pieces of mixed substance, on the other hand, ought then to be recrushed, finely enough to liberate substantially all of the magnetite from adhering gangue, and then re-separated, and the products of this re-separation added to the concentrate and tailings of the primary coarse separation. It

will thus only be necessary to crush coarsely the whole mass of ore at first; the concentrate and tailings then obtained comprising about nine-tenths and the middlings about one-tenth of the mass; and the impairment of the concentrate on the one hand, or the loss of valuable material in the tailings on the other, will be avoided. It should be borne in mind that the middlings, corresponding closely in quality to the average of the crude ore, would represent the same value, after the first coarse separation, as the cost of mining and crushing to the same condition of an equal quantity of the crude ore plus the cost of re-crushing and re-separating, and that but a trifling expense would be involved in the further reduction of the small percentage of these middlings to the fineness required to liberate the magnetite and to complete their final clean separation into heads and tails.

A consideration of the following analysis of the distribution of products derived from ten crude units by this system, and by variations therefrom, will show the importance of a close adherence to the methods of separation here prescribed. From equation No. 1 and the preceding analysis it was found that 1.879 units of crude ore (one-half magnetite and one-half gangue), if properly treated, could make 1 concentrate unit, consisting of 90 per cent. of magnetite and 10 per cent. of gangue, and 0.879 unit of tails, consisting of 4.5 per cent. of magnetite and 95.5 per cent. of gangue; hence the concentrate would constitute 53.22 per cent. and the tailings 46.78 per cent. of the crude ore thus treated. Therefore:

				Per cent.	Units.	Units.
10 units of crude ore =	9 units =	Concentrate, 53.22 per cent. =	Magnetite,	90.	= 4.311	
			Gangue,	10.	= .479	
						4.79
		Tailings, 46.78 per cent. =	Magnetite,	4.5	= .19	
			Gangue,	95.5	= 4.02	
						4.21
	1 unit of middlings, same as crude, re-crushed and re-separated. =	Concentrate, 53.22 per cent. =	Magnetite,	90.	= .47898	
			Gangue,	10.	= .05322	
						.5322
		Tailings, 46.78 per cent. =	Magnetite,	4.5	= .021051	
			Gangue,	95.5	= .446749	
						.4678
			Total,			10.

the sum of concentrate units being 5.3222 and of tails, 4.6778; total, 10.

If, however, instead of re-crushing and re-separating the middlings, they should be added to the concentrates, the result would be:

			Per cent.	Units.	Units.
10 units of crude ore =	Concentrate, 57.9 per cent. =	Magnetite,	83.09	= 4.811	
		Gangue,	16.91	= .979	
					5.79
	Tailings, 42.1 per cent. =	Magnetite,	4.5	= .19	
		Gangue,	95.5	= 4.02	
					4.21
		Total, ...			10.

Thus, while increasing the percentage of concentrate obtained from the crude from 53.22 per cent. to 57.9 per cent, which would add 0.468 unit to the product from 10 units of crude, the gangue in the concentrate would be increased from 10 per cent. to nearly 17 per cent., and the magnetite lowered from 90 per cent. to 83.09 per cent., which would correspond to 83.09 per cent. magnetite $\times$ .7242 $=$ 60.17 per cent. iron—the percentage of waste in the tails remaining the same as in the former case.

If, on the other hand, to avoid this depreciation in quality of the concentrate, the middlings should be thrown out with the tails, the result would be:

			Per cent.	Units.	Units.
10 units of crude ore =	Concentrate, 47.9 per cent. =	Magnetite,	90.	= 4.311	
		Gangue,	10.	= .479	
					4.79
	Tailings, 52.1 per cent. =	Magnetite,	13.24	= .69	
		Gangue,	86.76	= 4.52	
					5.21
		Total, ...			10.

From which it appears that 4.311 units of magnetite in the concentrate $\div$ 5 magnetite units in the crude $=$ 86.2 per cent. of the magnetite in crude saved in concentrate, and 0.69 $\div$ 5 $=$ 13.8 per cent. lost in tails; and also, by equation No. 1, that $\frac{90-13.24}{50-13.24} = 2.0881$ crude units to make 1 concentrate unit, instead of 1.879 required in the first case.

As will now be clearly apparent, any departure from the three-part method of separation—above outlined and analyzed, and by which a coarse concentrate of high purity would be obtained—necessarily involves one or another of three alternatives: either the great expense of more finely crushing the whole mass of ore, in order to enable an efficient two-part separation to be made, with the added difficulties and expense attending the transportation and use of such fine material in the furnaces; or a great reduction, by admixture of the middlings, in the usefulness and value of the concentrate; or a fatal reduction in the efficiency of the operation, by passing the middlings into the tailings.

The difference between concentrates containing 60 and 65 per cent. of iron is really of great importance; for, as has been shown, while the latter would contain only 10 pounds of gangue to the hundred of concentrate, the former would contain nearly 17 pounds. Concentrates carrying 17 pounds of gangue to the 100 would, in most instances, by reason of containing, also, too much phosphorus and sulphur, fall outside of the Bessemer class of ores; and, furthermore, it has been found that a difference of 5 units of iron to the 100 in the burden of ore fed to a furnace will, if properly controlled and utilized, make a difference of about 20 per cent. in the output of iron from the furnace.

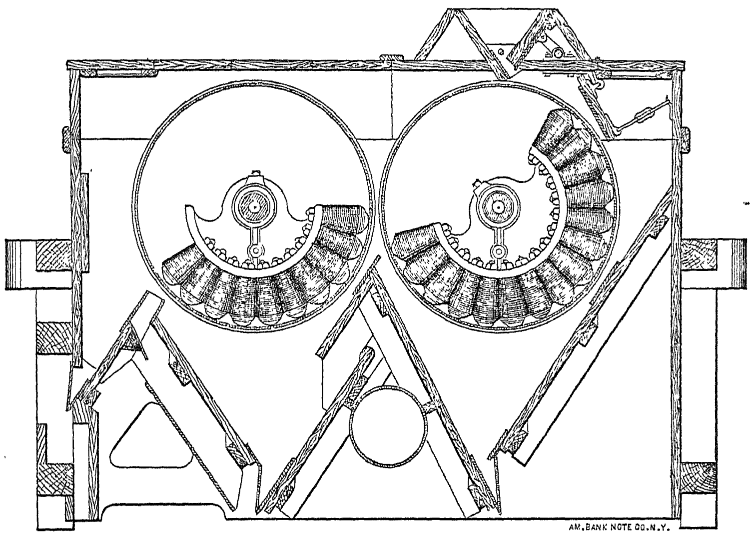
To execute successfully the three-part method of separation above outlined, an apparatus capable of working with great precision will be required. It must have the capacity to discriminate accurately between the several grades of material under treatment, to make a clean selection of magnetic from wholly non-magnetic substances, and among the magnetic particles of various degrees of purity and magnetic susceptibility, to differentiate the middlings from the pieces of pure magnetite. All this has been accomplished by the construction shown in the accompanying illustration, which presents a longitudinal, vertical section of the latest form of the Ball-Norton separator for iron-ore, and in which a complex field of magnetism, centrifugal force, gravity and the resistance of a counter-current of air, moving opposite to the direction of the ore-travel, are brought into coincident action to effect the required ends.

The distinguishing features of this apparatus, which having

been elsewhere more particularly described,* will be but briefly referred to here, are:

1. A stationary range of magnetic poles of alternately opposite polarity in the direction of the ore-travel, underneath which the drums, enclosing the two groups into which the range of poles is divided, may be rotated and may serve as carriers of the granulated ore, the iron particles being held upon the under side thereof by magnetic attraction.

FIG. 1.



The Ball-Norton Electro-Magnetic Separator.

2. Means for applying a strong counter-current of air to the moving mass of ore while it is suspended upon the under side of the rapidly-running drums and being driven along through the machine.

3. Provisions for differentiating the ore under treatment into the three grades which have been spoken of, this being effected by a differential speed of rotation of the two drums, assisted by relative adjustment of the strength of magnetism in the two groups of alternating magnets.

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* *Trans.*, xix., 187, "The Ball-Norton Electro-Magnetic Separator;" also United States Patents 404,332, 404,333, 404,334 of May 28, 1889, 430,058 of June 10, 1890, 449,726 of April 7, 1891 and 478,551 of July 12, 1892, to Ball and Norton; and Patent 465,349 of December 15, 1891, to C M. Ball. The patents here referred to are owned, and the commercial introduction of the separators is controlled and conducted, by The Magnetic Separator Company of Troy, N. Y.

Among the noticeable and important functions performed in the operation of this machine is that of tumbling or rotating the magnetic fragments of the ore, while they are suspended upon the underside of the drums, by their passage through the range of magnetic fields of successively opposite polarity; and it is upon this feature as a foundation that the striking development of magnetic separation since the year 1888 has been built. Prior to that time the various conceptions of magnetic ore-separators which had been embodied in the hundreds of patents granted in different countries during a period of forty years, contained one characteristic feature common to them all and contributing to their inefficiency. In every case, whatever devices may have been employed for moving the ore through the machine, the magnets were so arranged as to present a field of unchanging polarity to the ore in the direction of its motion. The attractive power of the magnets, combined with means for moving the ore, simply served to carry along such portions as were attracted, there being no possibility of separation going on after the mass had once become attached to the carrier. Moreover, in many cases the arrangement was such that the ore was carried over instead of under the magnets.

It is now well understood, that when a mass of sub-divided ore, consisting of mingled magnetite and gangue, is brought within the influence of a pair of magnetic poles, the grains of magnetic material unite in the formation of continuous loops, extending from one magnetic pole to the other in the direction of the lines of force, and that such loops will always, in their first formation, mechanically impound and pick up many of the gangue-particles; and it is clear that this impounded gangue can only be got rid of by breaking up the formation of these loops, while an adequate force is at the same time acting to remove the gangue-particles, as soon as they may be released from the grip of the attracted magnetite. This needed breaking up of the loops of magnetite and removal of the gangue is effected, in the machine illustrated, by tumbling or rotating the magnetic particles, end over end, by passing them through the successive fields of alternating magnetic polarity; while gravity, centrifugal force and the counter-current of air are acting coincidentally to remove the non-magnetic substances.

It will be seen that in this system two of the forces brought

into action are susceptible of adjustable regulation, viz.; the centrifugal force (by variations in the speed of the drums) and the magnetism. In practice it has been found that adjustment in the strength of the second group of magnets, in the line of the ore-travel, is capable of controlling with the utmost precision the selective power of the apparatus over the particles of pure magnetite and the middlings; so that particles of middlings below a certain predetermined degree of purity may be shaken out, during the tumbling or rotative action to which the magnetic particles are subjected in the alternating magnetic field. That is to say, it can be provided, by such adjustment, that particles consisting of five-eighths magnetite and three-eighths gangue shall be retained with the purer particles in the concentrate, while particles carrying more than that proportion of gangue will be thrown off in the middlings; or the division may be made on the line of any other desired proportion. Thus the grade of purity of the first concentrate and the percentage of middlings to be re-crushed and re-separated can be accurately controlled. There was no such discriminating power in the earlier types of separators; and, although a part of the ore obtained from a first separation might again pass through a second and sometimes a third separation, these machines were all, essentially, only two-part separators; for, at each passage of the ore, only two parts were produced, neither of which could be economically disposed of at once, as finished.

The impression has been somewhat prevalent among iron-makers that the cost of crushing and separating ore would be prohibitive, but this impression has doubtless been founded upon the poor results obtained from attempts to use inefficient and incapable types of separators and from faulty conceptions of method. Two-part separation with the old types of separators, water-jigs, etc., could find but one end—a desperate and costly struggle against discouraging results; poor concentrates, or excessive outlay in plant and from 20 to 40 per cent. of the value contained in the ore left in tailings. On the other hand, as has been shown, a separating-plant organized to execute a correct method may easily work at an efficiency of above 95 per cent., and this standard, in commercial practice, has been reached in operations extending through some years of time.

Fabulous sums are expended in mining, transporting and melting slag-making rock in furnaces. A modern furnace costs much more than a modern separating-plant, both in original outlay for construction and in expense of operation, and can not begin to handle and dispose of so much råw material; and while the gangue or slag-making material when treated in the furnace must be transported, handled, melted, fluxed and otherwise disposed of, if brought to the separator this rubbish is, at once, thrown out at small expense and left on the tailings-dump at the mine; and those charges are saved. It is a good-sized furnace which, working upon and fluxing out impurities from a 50-per-cent. ore (until recent times considered a good crude ore), can make an output of 100 tons of pig-iron per day, and its expense of operation is great. But a sufficient amount of 65-per-cent. concentrate to feed such a furnace can be easily made by a single separator of the type here illustrated; such a separator will handle from 16 to 25 tons of crude ore per hour, and produce therefrom 8 to 15 or 16 tons of concentrate; the amount of crude which can be treated and the output of concentrate depending upon the richness of the ore and its condition, whether fine or coarse. It is manifest, therefore, that the separator could more than feed the furnace; but, as already noted, the use of the richer ore would result in an increase of output, enabling the furnace to make about 165, instead of 100 tons of iron per day, and thus absorb the product of the separator. Thus a furnace and a separator may have, under these circumstances, nearly the capacity of two furnaces; and a comparison of cost of construction and operation of plant would be very greatly in favor of the separator.

Uniformity in quality of the concentrate is a natural result of the normal operation of the separator, which also leads to uniformity in quality of product from the furnace. This is of great consequence in steel-making; for while it is true that large supplies of rich ores may be obtained from Lake Superior mines and other sources, they are not of uniform grade and purity; and considerable variations in percentages of objectionable impurities will be found in successive lots of ore from the same mine. On the other hand the magnetic concentrate from a particular mine will always be practically uniform, whatever the variations in the crude ore from which it is extracted; and

its richness will be equal to that of the very best of native ores. The separator, therefore, is an available means to aid the manufacture of better, as well as cheaper, iron.

Since 1888, many plants for crushing and separating iron-ore have been projected, and a considerable number have been put in operation. Among the first in point of time and absolutely first in the technical and commercial success of the operation was the plant of the Magnetic Iron Ore Company, at Benson Mines (formerly known as Little River), St. Lawrence county, N. Y. At that place, for the first time in the history of the art, magnetic separation was successfully resorted to on a commercial scale, as a means of treating a low-grade, non-Bessemer ore to eliminate deleterious impurities and render it fit for steel-making.

Prior to 1889, the Magnetic Iron Ore Company had expended a large amount of money in building a railroad to the mines at Jayville, N. Y., developing them and securing the property at Little River, before it was finally discovered that ore of good quality, in its natural state, could not be taken from the mines in paying quantities. It was therefore decided by the company that the only thing to be done, to save the investment in these properties, was to establish a large concentrating plant at Little River—now Benson Mines—and to extend the railroad to that point. There exists, at that place, one of the largest known deposits of low-grade magnetite in this country. The ore is of a dense, fine-grained structure, averaging 45.5 per cent. magnetite, corresponding to 32.95 per cent. of metallic iron; and the crystals of magnetite, which approximate $\frac{1}{16}$ inch in average diameter, are associated in the formation with small crystals of apatite and pyrites. These characteristics of the crude ore render the problem of milling and concentration, from both the technical and commercial point of view, a most difficult one; for, in order to find a market, the ore must all be concentrated, and, to reduce the sulphur and phosphorus to limits which will fit the product for steel-making, the iron must be raised in the concentrate to run uniformly above 64 per cent., or to carry, say, 88.5 per cent. of magnetite and not more than 11.5 per cent. of gangue. To reach this result without a commercially fatal loss of iron in the tailings is an additional difficulty. The problem was solved, however, after a

close study and careful consideration of all the then known means of concentration, by an application of the Ball-Norton system of magnetic separation, which was put in successful operation very early in the year 1889.

The plant of this company, which was greatly enlarged during the year 1890, was, after that time, maintained in nearly continuous operation until the great depression in the iron-business of July, 1893. The works have a capacity, when in full operation, to crush and separate one thousand tons of crude ore per day of twenty-four hours, yielding about five hundred tons of concentrate; the primary crushing of the ore being carried to $\frac{1}{14}$ inch, and the average character of the crude ore and products being as follows:

	Magnetite. Per cent.	Metallic iron. Per cent.	Sulphur. Per cent.	Phosphorus. Per cent.
Crude ore,	45.5	32.95	1.00	.15
Concentrate,	88.5	64.09	.21	.032
Tails,	4.	2.90		

From these factors, we find, according to equation No. 1, $\frac{88.5-4}{45.5-4} = 2.0361$ crude units to make one concentrate unit, and, by equation No. 3, $\frac{88.5 \times 100}{2.0361 \times 45.5} = 95.53$, the per cent. of the magnetite in crude saved in concentrate, 4.47 per cent. being lost in tailings; thus realizing the conversion of a low-grade, non-Bessemer into a high-grade Bessemer-ore, with an efficiency of 95.53 per cent. in a large commercial operation.

Although these mines are somewhat remote from the great consuming points of the country, and are therefore, to some degree, hampered by the heavy cost of transportation to market, this unfavorable condition is largely offset by the enormous extent of the ore-body above drainage-level and the consequent cheapness of mining, and by the perfectly uniform and desirable quality of ore obtained as a product of concentration. The owners of the mines deserve great credit, as well as profit, for the courage and liberality with which they attacked a new and difficult problem and entered as pioneers upon the commercial development of a new art; and doubtless, with a return of activity in the iron-business, they will have their reward.

Following closely upon the operations at Benson Mines, there

was developed throughout the country a great revival of interest in the subject of magnetic ore-separation, and many plants were projected and undertaken in different places.* At Mineville, Essex county, N. Y., Messrs. Witherbee, Sherman & Co. carried out, during the years 1889–92, a series of elaborate and costly experiments with various methods and tests of many different types of separators, finally adopting the Ball-Norton system and machines. At Bechtelsville, Pa., Dr. H. K. Hartzell and Mr. Thomas A. Edison were experimenting with a small plant in 1889, which was unsuccessful and abandoned; and a little later Dr. Hartzell began operations at Rittenhouse Gap, Pa., where, after experimenting with different methods and machines, he finally, in 1890–91, installed the Ball-Norton system. After being for some time in successful operation, this plant was, unfortunately, burned down in 1891, but was immediately rebuilt, and its operation was resumed in 1892.

In addition to the above, which have been thus mentioned on account of their historical significance in connection with the development of the art of magnetic ore-separation, license contracts have been made with various persons and corporations in different parts of the country by The Magnetic Separator Company of Troy, N. Y., for the use of the patented features of the Ball-Norton system; among others, with the Chateaugay Ore and Iron Company, of Plattsburgh, N. Y., whose mines are at Lyon Mountain, Clinton county, N. Y., where there now exists one of the largest ore-crushing plants in the country—having a capacity of about 1200 tons of crude ore per day; also with the owners of the Arnold Hill mines at

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* For fuller particulars respecting many of these plants, descriptions of magnetic ore-separators of the different types, papers on ore-dressing, and discussions connected with the development and progress of magnetic separation, see *Transactions*, vol. xvii., 1889, and volumes subsequently issued to date; also, an interesting and valuable contribution to the literature of the art by Axel Sahlin, M.E., in the *Engineering and Mining Journal*, in three parts, June 11, 18 and 25, 1892, "Introduction and Development of Magnetic Separation of Iron-Ore;" also the *Report of the Mining Inspector of the State of New York for the Year* 1893. See also the literature of suits in the United States Circuit Courts; The Magnetic Separator Company *vs.* International Ore-Separating Company, in the District of New Jersey, and the Magnetic Separator Company *vs.* William D. Hoffman, in the Southern District of New York. The records of these suits are in the possession of Mr. Charles L. Buckingham, 195 Broadway, New York City, counsel for complainants.

Ferrona, Clinton county, N. Y., and with the Tennessee Coal, Iron and Railroad Company. This latter company is at the present time prosecuting, at Bessemer, Ala., a valuable and interesting test in the concentration of red, fossiliferous ore by first roasting, to render the ore magnetic, and then concentrating with magnetic separators. The outcome of this operation will be watched with great interest by southern iron-masters.

The coarse-grained magnetites of the Champlain district of the Adirondack region, carrying more iron in the crude than the Benson Mines ore, yield especially satisfactory results in concentration, on account of the general facility with which they may be crushed and prepared for separation, and the coarse-grained character and high purity of the product. A typical example may be taken from results obtained in the rough commercial treatment of ore from the New Bed mines of Witherbee, Sherman & Co, at Mineville, N. Y., primary crushing carried to $\frac{1}{6}$-inch, and the average character of crude ore and products being as follows:

	Magnetite. Per cent.	Metallic Iron. Per cent.	Phosphorus. Per cent.
Crude Ore, ...	66	47.8	0.7
Concentrate, ...	94	68.075	.0123
Tails, ...	8	5.79	

From which, by equation No. 1, $\frac{94 - 8}{66 - 8} = 1.483$ crude units to make one concentrate unit, and, by equation No. 3, $\frac{94 \times 100}{1.483 \times 66} = 96.03$, the per cent. of the magnetite in the crude ore saved in concentrate, 3.97 per cent. being lost in tailings.

On account of the increase of capacity given to furnaces and the improvement in quality of product by the use of richer ores, a supply of high grade concentrates of uniform character is of nearly equal importance to makers of foundry pig-iron as to the makers of Bessemer iron and steel; and since results corresponding to those which have been cited can be obtained wherever similar ores are found, this puts at the disposal of all persons interested in the trade a means (where suitable conditions of application exist) of economizing and improving the manufacture of iron and steel.

At a moderate estimate, the ore-crushing capacity of exist-

ing plants, located in the Adirondack region of New York and in New Jersey and Pennsylvania, is sufficient to make therefrom more than 1,000,000 tons of high-grade concentrates per annum. Not all of these plants are, as yet, equipped with the most efficient types of separators, but, doubtless, with the advent of an increasing activity in business circles, deficiencies in this respect will be remedied in time, and separated magnetic ore will fill an important place in the supply of raw material for the manufacture of iron and steel.

Southern Magnetites and Magnetic Separation.

BY HARVEY S. CHASE, NEW YORK CITY.

(Atlanta Meeting, October, 1895.)

DURING the recent great depression in iron, little has been done in the magnetic separation of iron-ores; and previous to the present decade the art was in its infancy, so that there were few, if any, successful separators on the market, and but scanty practical data could be obtained as to the industry itself. Moreover, the recent development of the great Mesabi Bessemer beds, coincident with the intense depression in business, has discouraged active attempts to solve the problems presented in the crushing and separation of the refractory magnetites of the South.

Experiments have been carried on, however, upon a practical scale in separating these ores magnetically, at the well-known Cranberry mines, in Mitchell county, N. C., with results which are encouraging.

The magnetites of the Blue Ridge, in the South, have been known, and the soft ores of the region have been used in Catalan forges, since very early times. The original settlements in East Tennessee were made along the Watauga river by Daniel Boone, William Bean, John Sevier, and other stalwart pioneers; and this valley lies parallel with the "Cranberry" lead, which is undoubtedly the most celebrated and the most extensive deposit of workable magnetic ore in the Southern States.

This lead I have personally traced along the outcrop, be-

tween similar hanging- and foot-walls, for more than twenty miles. On the western portion of it, in Carter county, Tenn., is the notable iron property of the Watauga Associates, known as the "Magnetite Lands." Analyses of the ores from this tract will be found, among others, in the table appended to the present paper (page 556).

As this region is the one with which I am most familiar, my deductions will be based mainly upon results obtained from its ores, although similar deductions can be properly applied to the greater part of the Blue Ridge and Great Smoky magnetites, or at least to those carrying low percentages of titanium; for titanium is the curse of magnetite in the South, as elsewhere.

As has been pointed out in previous papers before the Institute, the Blue Ridge magnetites may be roughly divided into two belts, running in general direction parallel with each other and with the mountain crests; the eastern carrying mainly titaniferous, and the western non-titaniferous ore; although there are also well-defined leads of non-titaniferous ores parallel with the others, and lying still further eastward, as described in Mr. Nitze's paper on the "Magnetic Iron-Ores of Ashe County, N. C.," in 1892.*

In this paper I shall confine my attention mainly to the non-titaniferous ores. Even when free from titanium, and when carrying phosphorus and sulphur below Bessemer limits, these ores present no inconsiderable difficulties in the way of successful and profitable magnetic separation.

In the first place, it appears that few, if any, Southern magnetites can be profitably worked in which the "run-of-mine" carries less than 40 per cent. of metallic iron, and from which, by hand-picking, or by magnetic cobbing, a shipping-ore running from 45 per cent. upwards cannot be obtained, to the amount of at least one-half of the total mine-output.

This grade of ore, even if "special Bessemer," will commercially bear shipment only to neighboring furnaces; but the balance of the output, by judicious washing, cobbing, screening, crushing and separating, should be increased in percentage of iron to from 55 to 65 per cent. or even higher, and can then of course be shipped to greater distances.

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* *Trans.*, xxi., 260.

I do not mean to assert that there are no titaniferous magnetites in these regions which can be profitably worked. Local conditions, and the character of the ores in other particulars, make sweeping general statements unadvisable. But it can safely be said that, like phosphorus-bearing ores, the titaniferous ores would require crushing to comparatively fine mesh in order to mechanically break apart the small crystals of ore from the gangue, so that the latter may be magnetically eliminated; and we know that this fine crushing of large quantities is the very thing to be avoided in handling these refractory ores.

The experience with magnetic separation at Cranberry is noteworthy in this particular. Here we have an excellent Bessemer ore, non-titaniferous, comparatively cheaply mined, partly in open cut and partly in tunnel, with good transportation-facilities, and with a "run-of-mine" averaging about 42 to 43 per cent. of metallic iron. The greater portion of this iron is in magnetic oxide, Fe_3O_4, but from 5 to 11 per cent. is present as FeO in hornblende. Should the total output be crushed and magnetically separated, this FeO would be lost in the tails, and it becomes a question of considerable importance to decide when to save the hornblende and when to sacrifice it, since within these percentages of iron the margin of profit—taking wear and tear of machinery into consideration—may be located.

With the recent marked revival in the iron-trade, the question of profitably producing Bessemer pig with ores from the Cranberry district and coke from southwest Virginia, has once more assumed importance, and the element which will be found decisive in this matter (assuming that the coking-coals of Big Stone Gap are satisfactory, and that the necessary railroad-extensions will be made) is magnetic separation.

It is therefore interesting to examine the result of the experiments made at Cranberry, in 1892 and 1893, under the direction of Mr. Frank Firmstone, to whom I am indebted for the data here given.

As Mr. Firmstone's object in these tests was wholly a practical one, namely, to procure ore for the furnace at Cranberry from material which otherwise would have been thrown on the waste-dump, and to do this at a cost not greater than that of

mining an equal amount of new ore, he did not attempt to particularize the various elements of cost of crushing, washing, screening, and separating; and he expressly states that too great dependence should not be placed upon the tabulation of his results, as the practical running was for a comparatively short time and with crude and insufficient machinery. Nevertheless, the results attained are distinctly encouraging, and certain deductions may obviously be drawn.

The total cost of Mr. Firmstone's treatment of the waste ore averaged about 45 cents per ton of concentrates obtained, part of which carried 63 per cent. of metallic iron, and the average about 47 per cent., while the cost of mining an equivalent amount of new ore would have been at least 70 cents per ton. This cost of 45 cents per ton for cobbing, crushing, washing, screening and separating, covers all labor and materials, including repairs, except cost of power, which was derived from the furnace-boilers, and was nominal.

This cost, Mr. Firmstone says, could certainly be reduced one-half with a larger plant and improved machinery; and, making due allowance for power, we may consider 25 cents per ton as a safe figure for the cost of this separation per ton of concentrates produced; the cost of mining, of course, being charged against the shipping ore, and the material used for separation being considered as otherwise a waste product.

The question of importance then appears to be whether fine crushing shall be attempted, with the accompanying advantage of the higher grade of concentrate produced, but at greatly increased cost for power, for repairs and renewals, for fine screens and for comparatively delicate and discriminating magnetic separators (like the well-known Ball-Norton or the Chase machine, described before the Institute meeting at Plattsburg in 1892*), and with the production of a fine-grained furnace-burden and the necessarily large loss of iron as FeO in the tails.

This question is fundamental; and the experience with the Cranberry ores, while perhaps not decisive, strongly points in favor of a minimum of crushing and a maximum of coarse ore, even though it be of lower percentage of iron.

The ore at Cranberry, before shipping, is passed over bar-

* *Trans.*, xix., 187, and xxi., 503.

screens having $1\frac{1}{2}$-inch openings, and what goes through, along with any clay from the open cuts which shows much ore in small pieces, is washed in a log-washer, in which it is freed from clay and very fine rock. A revolving screen on the washer-shaft separates the washed stuff into four sizes, viz.: (1) coarser than $1\frac{3}{4}$-inch round hole, which is hand-picked; (2) between $1\frac{3}{4}$ and $1\frac{1}{4}$-inch round; (3) between $1\frac{1}{4}$ and $\frac{3}{16}$-inch round, which two sizes are treated separately on two Wenström separators; and (4) finer than $\frac{3}{16}$-inch round, which is washed by a stream of water to a revolving magnet. The heads from this magnet are again divided by a revolving screen (8 holes per inch, of No. 18 wire); the rejections, called "shot" below, going to the furnace, and that passing through ("dust") being sold.

As the fines screened from the shipping-ore have not generally furnished enough clean ore to run the furnace, the deficiency is made up by crushing, in a Buchanan crusher, mixed rock and ore thrown out from the shipping-ore and picked out from the old dumps. This material, after crushing, goes through the washer and separators, with the fine from the mines.

A test of the revolving magnet was made April 15, 1892, all the tails being caught and weighed up, and resulted as follows:

	Pounds.	Iron in Fe_3O_4. Per cent.	Iron in FeO. Per cent.	Total iron. Per cent.
Shot,	1092	49.8	4.9	54.7
Dust,	1014	61.0	2.4	63.4
Tails,	874	11.0	11.1	22.1

There should also be added to the quantity of tails the fine material lost in the wash-water, which amounts to from 5 to 40 per cent. of the whole material treated according to whether clean (tunnel) or dirty (open-cut) ore predominates.

From various tests of this wash-water with hand-magnets, it has been found that the loss of iron in it is negligible, and including the weight of the material in the wash-water would reduce the percentage loss in tails very materially. Inasmuch as half of the loss in tails is in non-magnetic FeO, we may consider the result, on the whole, as a good separation.

During the month of March, 1892, 920 tons of ore from the crusher and 947 tons of "dirt" from the mine, or a total of 1867 tons, were sent to the washer and separators. From this

Analyses of Representative Magnetic Ores from North Carolina and Tennessee.

	State and County.	Name of Mine.	Metallic Iron.	Magnetic Oxide.	SiO_2.	P.	S.	TiO_2.	CaO.	Al_2O_3.	MgO.	MnO_2.	Analyst.
1	Carter, Tenn.	Magnetite	63.63	......	......	0.036	......	......	......	......	......	......	
2	" "	"	66.39	......	......	0.0007	......	......	......	......	......	......	
3	" "	"	54.63	......	......	0.032	......	......	......	......	......	......	
4	" "	"	56.14	......	......	tr.	......	......	......	......	......	......	
5	" "	Wilcox	57.94	......	......	0.017	......	......	......	......	......	......	
6	" "	Fork Ridge	58.35	......	17.09	0.0071	......	0.007	......	......	......	......	Bidtel, '90.
7	" "	Magnetite	45.93	63.77	20.97	0.000	0.02	......	10.10	2.87	1.43	0.42	Riley, '90.
8	" "	Crab Orchard	50.77	......	17.39	0.0087	......	0.013	......	......	......	......	Bidtel, '90.
9	Mitchell, N. C.	Cranberry	58.49	80.77	9.08	......	......	......	......	0.52	......	1.42	Genth.
10	" "	"	66.53	91.89	4.02	tr.	0.025	......	1.06	1.03	0.23	0.32	Chandler, '69.
11	" "	"	68.34	94.37	4.16	......	......	......	0.43	0.42	0.36	0.26	
12	" "	"	66.22	91.45	5.74	......	......	......	1.01	0.77	0.53	0.06	
13	Orange, "	......	54.81	75.69	20.83	0.0187	......	......	......	......	......	......	Genth.
14	Catawba, "	......	57.50	82.14	4.47	......	......	......	......	......	......	0.53	"
15	" "	......	60.66	86.66	1.42	......	......	......	0.25	......	0.27	5.12	"
16	" "	Ormond	66.75	92.18	4.34	......	......	......	0.35	0.44	2.23	0.28	"
17	Yadkin, "	Hobson	51.83	71.68	24.62	tr.	tr.	......	0.57	2.46	0.10	tr.	
18	" "	"	57.75	79.75	14.46	0.0375	......	......	0.82	1.20	0.98	0.81	
19	" "	"	67.79	93.61	4.62	0.05	......	......	0.45	0.20	0.86	0.11	
20	" "	Rogers	57.13	79.71	15.66	......	......	......	0.31	2.27	0.17	tr.	
21	" "	"	65.34	92.47	7.20	......	......	......	0.13	tr.	0.20	tr.	"
22	Caldwell, "	Richlands	67.32	......	2.25	......	0.01	......	......	.87	......	tr.	Hanna.
23	Ashe, "	McCarter	51.95	......	5.37	0.018	......	9.17	......	......	......	......	McCreath.
24	" "	Young	52.85	......	4.35	0.013	......	8.800	......	......	......	......	"
25	" "	Helton	58.93	......	11.075	0.033	0.068	......	......	......	......	......	
26	" "	McClure's	56.00	......	......	0.013	0.05	tr.	......	......	......	......	C. B. White.
27	" "	Ballou	65.40	......	3.20	0.011	......	......	......	......	......	2.58	"
28	" "	"	65.65	......	0.80	0.004	......	......	......	......	......	3.83	McCreath.
29	" "	Horse Creek	64.58	......	4.12	0.011	......	......	......	......	......	2.21	White.
30	Guilford, "	Shaw	57.68	79.53	0.76	......	......	13.52	0.45	1.68	2.79	0.81	Genth.
31	" "	"	52.68	72.74	5.68	......	......	11.67	0.76	5.08	2.61	0.64	"
32	" "	"	55.06	76.04	1.30	......	......	13.60	0.60	4.26	2.33	0.96	"
33	" "	"	54.17	74.81	1.80	......	......	14.46	0.69	2.66	3.09	1.53	"

was recovered 1029 tons of "clean ore," averaging 45 per cent. of metallic iron, 47 tons of "shot" averaging 55 per cent. of metallic iron, and 113 tons of "dust" averaging 63 per cent. of metallic iron; also 271 tons of tails and 406 tons of fines washed away (by difference) averaging together less than 20 per cent. of metallic iron.

During 1892, therefore, with the original and comparatively crude machinery, about 1100 tons of "shot" and "dust" of these high percentages of iron were saved from the waste of the mines, besides a much greater amount of the separated "clean ore" (larger than $\frac{3}{16}$ inch).

It will be appreciated that the problem presented by these ores, already Bessemer, with only silica to contend with, is very different from that of many eastern ores carrying apatite or pyrite. The low percentage of iron and the large percentage of silica render the southern magnetites costly to crush; but, as we have seen, it is fortunately not necessary to crush fine in order to eliminate sufficient of the silica to make the product marketable locally under normal conditions and prices.

From these experiments, therefore, we may draw two apparent conclusions:

1st. That emphasis should be laid upon careful washing, screening and sizing rather than upon fine crushing; and

2d. That each size of material should be concentrated separately upon suitable magnetic separators, giving as a total result a Bessemer concentrate averaging from 50 to 60 per cent. of iron and of a size sufficiently coarse for furnace-burden without other mixing.

A table of representative analyses of Southern magnetic ores is given on page 556.

Onyx-Marbles.

BY PROF. COURTENAY DE KALB, SCHOOL OF MINES OF THE UNIVERSITY OF MISSOURI, ROLLA, MO.

(Atlanta Meeting, October, 1895.)

THE following observations upon onyx-marbles are fragmentary, and may shortly be rendered superfluous by the appearance of a work upon these interesting stones by Prof. George

P. Merrill, which is now in the press. The notes in this paper are the result partly of individual investigation, partly of information furnished by engineers and others who have studied onyx-marble deposits, and partly of research in the scanty literature upon this subject.

In the beginning a sharp distinction must be drawn between the precious onyx, which is a cryptocrystalline variety of quartz, and the ordinary commercial "onyx," which is a deposit of carbonate of lime from aqueous solution. The true, or precious onyx, is distinguished arbitrarily from the agates by the perfect parallelism of the color-bands, these bands consisting usually of alternations of white and black, white and brown, and white and red. It may be mentioned, in passing, that such perfect banding is so exceedingly rare that very few, if any, of the onyxes or cameos sold in our jewelry shops are from naturally-colored stones, the artificial coloring of agates being a regular industry in Germany. The method is said to consist in saturating the more porous layers of the banded white or bluish slate-colored agate with honey, and then carbonizing this with sulphuric acid, to produce the black-and-white variety. The red-and-white is produced by soaking in ferric chloride and precipitating ferric oxide with ammonia. Such a red coloration should consequently be obtained in any porous stone by similar treatment; but my personal experiments in this direction have been unsuccessful, or, at best, have resulted in a dirty ferric-oxide stain.

The term onyx-marble, as applied to calcareous deposits, must be still further limited, since many varieties exist. The general name of travertine will include all such deposits except the finely-crystallized minerals calcite and aragonite. The oölites should also be excluded from this classification, although in their manner of formation these more nearly approach the true travertines. The familiar calcitic formations in caves (stalactites, stalagmites, "cave onyxes") may quite properly be classed among the travertines. That which entitles any of these to be called an onyx-marble is the accidental circumstance of texture and beauty, fitting it to serve as an ornamental stone in decoration. It is, therefore, a commercial, and not a scientific, distinction.

The requisite qualities for a commercial onyx-marble are:

First, perfect, or nearly perfect, homogeneity of texture; second, absence of subcrystalline structure, so that no tendency to crystallization may be observable by the eye; third, freedom from porosity and cracks (although slight porosity may be corrected by "filling," and cracks, if not so deep and extensive as to weaken the stone, may often be highly colored and produce an acceptable artistic effect);* fourth, translucency, the *sine qua non* of a high-grade onyx-marble, giving a deceptive appearance of "depth;" fifth, beauty of coloring—a matter of taste and fashion, for the most part, although the translucent white, delicate mignonette green, and fine, translucent white, with dashes or veinlets of pink, are almost always in demand and bring the highest prices; and sixth, proper size of perfect blocks, the lowest limit for thickness being 1 inch, although slabs three-fourths to one-fourth of an inch thick are sometimes used, with a "backing" of other material, while for superficial area the line is drawn at 1 foot square, although here, again, smaller sizes, if very fine in color and texture, may be marketed.

The size of rough blocks, as worked out at the quarry, must necessarily be larger than that of the perfect blocks or slabs finished at the factory. The effect of the tool in "pointing," shatters or "stuns" the stone to a depth of one-half inch or more. The factor of waste in rough blocks, as they are at present sent to market, may be said to vary from one-third to two-thirds, and exceeds this in diagonal sawing for fancy effects.

The translucent white onyx-marbles are very often confounded with, and sold under the name of alabaster, the true alabaster being a translucent variety of gypsum, and far less durable, owing to its greater softness, than onyx-marble. Again, we find in commerce a stone called "agate onyx," which is a variety of onyx-marble containing more or less foreign matter, chiefly alumina, and sometimes silica, approaching the agates in appearance, but generally inclining, in part or in whole, towards opacity. While being highly ornamental, particularly in connection with dark wood interior finishing, its application is more limited than that of the finer, translucent varieties.

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* The cracks in onyx-marbles which injure their value are those usually of recent origin, which have not subsequently been filled with carbonate of lime. Old cracks which have been filled with cementing material almost invariably add to the beauty and value of the stone.

Before leaving the commercial side of this matter, a few additional details may prove of some importance. The highest prices are obtained by a combination of desirable physical characters with large size of blocks or slabs. The poorer grades bring sometimes as little as 50 cents per cubic foot; onyx-marble, in the rough at least, being invariably sold by this unit of size, whether in blocks or sawed into slabs. From this minimum, prices range upwards to $50 a cubic foot; and extremely fine blocks, suitable for columns, may command fancy figures, limited only by the size of the purchaser's purse, and the vending genius of the dealer. It may be said, however, that the average size of good slabs is only 12 by 14 inches, and that slabs 18 by 36 inches, and 2 by 4 feet in size are not uncommon. The famous Mexican variety known as Pedrara, very little, if any, of which is now in the market, is quoted on an average at about $10 per cubic foot. The "New Pedrara," running chiefly into the pink shades, brings about the same price. Fine translucent Arizona stone sells at about $8; California blue-and-yellow, at $6; other varieties, according to their fineness. In one large representative shipment from Vera Cruz, taken as an example, the blocks ranged from 36 by 16 by 13 inches to 53 by 33 by 24 inches, and were billed at $4 per cubic foot f. o. b. at Vera Cruz.

The cost of sawing and polishing varies from 65 cents to 80 cents per square foot, according as the polishing is done by the machine- or the hand-process. The sawing differs in no wise from that employed with other stones. The machine process is as follows: After being sawed, the slabs are placed on a "rubbing-bed," which consists of a circular cast-iron plate, from 8 feet to 15 feet in diameter; the older forms having a circular opening, from 1 foot to 18 inches in diameter, in the center. The plate is planed to a smooth surface, and is mounted upon running gear so that it may revolve in a horizontal plane. Fixed arms, usually four in number, are sustained radially about one-fourth of an inch above the plate, either by an upright passing through the central opening, or by a framework overhead (in the case of the newer solid forms of bed). The slabs of stone to be polished are placed upon the bed in front of the arms, and the bed is revolved slowly beneath them in such a direction as to hold them firmly against the arms. An abrading material, such as sand, sometimes mixed with "chilled shot" or crushed

steel, with a constant supply of water, is fed upon the plate. If necessary, the stones are weighted to increase the friction. From this rubbing-bed the slabs are removed to the emery-bed, which is similar to the former, fine emery being used for abrasion. They are then rubbed down by hand with a fine, evenly-grained sandstone, commonly called a "Scotch hone," with a sufficient supply of water, and smoothed off with pumice-stone and water. The final polish is put on by rubbing the slabs upon a buffing-bed, similar in form to the rubbing-bed, but covered with a thick, specially prepared felt, upon which a small amount of "putty powder" (oxide of tin) is fed, to give a high gloss. The hand-process consists in grinding on the rubbing-bed as before, and then rubbing down by hand successively with Nova Scotia "blue stone," "red stone," "Scotch hone," and pumice-stone, after which it is glossed with "putty powder," or, in the case of cheaper "onyxes" and common marbles, with a mixture of two parts of oxalic acid and one part of tin oxide. This latter finish produces a sort of "skin-coat," which, upon fracture, looks as if the stone had been varnished. The edges of onyx-marble table-tops, mantels, etc., are treated in this manner, even when the surface has been polished on machines. The use of common emery with white stones is objectionable, owing to its tendency to discolor them.

The amount of onyx-marble produced in the United States at present is very small, the reported output for 1893 having been 2175 cubic feet, which dropped in 1894 to 1450 cubic feet, averaging, at the place of production, $20 a cubic foot, showing that only the finest grades were marketed. The principal localities in this country where important deposits exist are California, Arizona and Utah. In California, the principal locality is in San Luis Obispo county, near Musick, in the Santa Lucia mountains. Here the inclosing rock is sandstone, the onyx-marble occurring in nearly vertical ledges 16 inches wide. The colors are white, with veins and "clouds" of red and smoky-black or blue. Blocks 10 feet square are said to be available. In Solano county deposits occur near Suisun, Vacaville, and elsewhere. In San Bernardino county occurs a light-brown variety, and an emerald-green shade is reported from Siskiyou county, along with others from Soda Springs and Yreka. A ledge 12 feet thick is found 25 miles from Santa Ana, in Los

Angeles county, and more or less is known to exist in Kern, Placer and Tehama counties. Almost without exception, in California, a close connection can be traced between the onyx-marble deposits and hot springs, or other mineral deposits known to have resulted from such waters. In Siskiyou county they occur along with hot springs which are depositing both onyx-marble and porous travertine. Eruptive rocks also abound in their vicinity. At the Suisun marble quarries is a breccia of shale, sandstone and volcanic ash, cemented by lime, and traversed by veins and bunches of aragonite (?).

In Arizona quite similar conditions obtain. The chief deposit is on Big Bug creek, in Yavapai county, 25 miles southeast of Prescott. It is a surface-formation, occupying a series of rounded knolls several hundred acres in extent, and is found in layers varying from a fraction of an inch to several inches in thickness, interbedded with a coarse breccia of schistose, granitic, syenitic, and dioritic fragments, cemented together by a sandy calcareous matrix. The country-rock is also schistose, granitic, and dioritic. The onyx-marble deposits themselves consist of irregular concentric layers, thinning out unevenly, with compact layers frequently separated by porous ones. The colors are variable. Some of the finest reseda-green onyx-marble in the world comes from these quarries. Their beauty is enhanced often by a pecular wavy effect of alternating light and dark shades of green, but such colors are rarely uniform throughout large blocks. Amber, ocher-yellow, and white masses are found; but the characteristic of these Arizona specimens, which is sure to appear in a slab of any considerable size, is a brilliant ochery red, running into a perfectly opaque chocolate brown, constituting the variety known as "agate-onyx." The more highly-colored specimens often yield as much as 5 per cent. of basic ferric carbonate. This changes to the hydrated sequioxide, producing the brown shades, and destroying the compact structure of the stone.

At Cave creek, Arizona, is another deposit of like character as to formation and colors. One ledge is 10 feet thick, but has been shattered by earth-movements. This is on the slope of a low hill, capped with basalt. The country-rock consists of schists, with dikes of acid eruptives. Throughout this region are large areas of lava *mesa*, underlain by volcanic tufa. These

yield the calcareous waters, which also contain more or less sodium sulphate. It is also worthy of note that, in the Eureka mining district of Arizona, there are other deposits of travertine below similar beds of tufa and lava.

Utah is becoming a producer of onyx-marble, with prospects of increasing in importance. The Salt Lake City Onyx Company is operating dressing-works in Salt Lake City, obtaining its stone from quarries to the west of Utah Lake. Directly above and in contact with the onyx-marble is a blue limestone. The deposit rests upon clay, sand, and limestone. There are many evidences of earth-movements, and the range in which the deposit occurs abounds in metalliferous veins. Six miles distant there is a hot spring issuing upon the surface at a temperature of 105° F. The predominating color of this onyx-marble is orange, but green, pink, lemon, and other shades are procured. Slabs measuring 10 feet 6 inches by 5 feet 8 inches have been taken from these quarries and finished up. Sizes from 12 by 18 inches to 12 by 36 inches can be obtained in considerable quantities.

Deposits are also reported from the vicinity of Fillmore, Millard county, ranging in color through lemon, orange, mahogany, and black. The onyx-marble occurs mostly associated with limestones and quartzites, along a belt of warm springs, running through Millard, Beaver, and Iron counties. These springs occupy mainly a line of contact with eruptive rocks.

A fibrous, concretionary variety of onyx-marble occurs near Rio Puerco in Valencia county, New Mexico; and a similar deposit is reported from El Paso, Texas. This description would seem to place them among the "cave-onyxes," concerning which much is heard in nearly all of the great limestone-bearing States in the Union. These are merely stalactites and stalagmites, and in some cases masses of compact travertine, forming incrustations upon the walls and floors of caves. The only one of these deposits now known to be in process of exploitation is that owned by the Eureka Onyx Company, of Eureka Springs, Arkansas, situated in the northeast part of Carroll county, near the Missouri line. The company is operating works in Eureka Springs, producing mostly small slabs, although mantel facings 12 by 24 inches in size, are also worked out. These sell when finished at from $1.00 to $2.50 per square foot. The colors are chiefly

white, with occasional tinges of red and pale green, rarely translucent, and often displaying the radiated fibrous structure so common in stalactites and stalagmites. Missouri has also produced small amounts from caves in Crawford and Pulaski counties. Sound blocks of large size, however, are infrequent, and efforts to work these deposits have proved unsuccessful. The colors are white and brown, varying from opaque to subtranslucent. Virginia has also yielded a small amount of this variety of onyx-marble, coming from the quarries of the Virginia Onyx Company, in Rockingham county. This locality, from the insufficient accounts obtainable, would appear to offer peculiarities worthy of further investigation. There is reported to be one considerable mass of compact travertine, covered with *débris*, in which occur a large number of detached masses of the same material, one of which is of important dimensions, standing nearly vertically. Whether or not this is the result of a collapsed cave remains undetermined. In Missouri, some of the largest masses of "cave-onyx" are found thus in the *débris* of ancient caves which have fallen in. Some of these caves had been of enormous extent, so that deep ravines and very considerable valleys occupy the lines of the ruined caverns. Stalagmitic bosses may be found high up on the hillsides along these ravines, and the "float" abounds in weathered fragments of stalactites, stalagmites, and calcareous incrustations. In the process of weathering, the banded structure of the incrustations becomes very prominent; the more opaque layers projecting boldly, while the clearer layers are worn away, and acquire a chalky appearance in the body of the mass. The Arizona onyx-marble weathers similarly, forming a finely-striated surface, with the opaque red and brown layers protruding.

By far the most important source of onyx-marble in the world to-day, is the republic of Mexico. The old localities are chiefly in the State of Puebla, between Vera Cruz and the city of Mexico. The famous "Pedrara" came from quarries near Tecali, twenty-one miles from the city of Puebla. Large blocks are no longer available there; but the manufacture of small ornaments by the natives is still an important industry in Puebla. Further to the southeast, in the district of Tehuacan, is the quarry known as Antigua Salines, where the principal deposits form the face of a hill 250 feet high. Thirty-five miles

west of Antigua Salines are the excellent quarries of La Sorpresa and La Mesa; the former yielding a semi-translucent to whitish stone, lacking, however, in the brilliancy which distinguishes the product from Antigua Salines. These deposits are either superficial or included between masses of siliceous country-rock, in the manner of veins. The old Tecali deposits are largely broken up, occurring in the form of boulders in a matrix of red clay, overlying conglomerate. The region has been much disturbed by volcanic agencies, and hot springs are abundant.

The largest onyx-quarries in the world to-day are those opened in 1892 by the New Pedrara Onyx Company of New York in the peninsula of Lower California. They are situated in a desert, 40 miles from the Pacific ocean and 2300 feet above its level. There are two series of deposits, three or four miles apart, the larger one showing outcrops over 20 acres. They have been formed in a shallow arroyo, or ravine, between flat-topped ridges of horizontal Cretaceous strata overlain, a few miles distant, by basaltic lavas. A recent writer in the *Engineering and Mining Journal* says:

> "Within the arroyo and immediately under and between the layers of onyx are soft limestones and conglomerates with lime cement, probably belonging to a series of Tertiary or recent beds deposited in an irregular lake that once filled a great interior valley which occupies the medial portion of the peninsula, parallel with its shores."*

Beneath these Tertiary deposits lie granites and gneiss. The onyx-marble was evidently deposited from the waters of warm springs, which extended in a line up and down the arroyo. Three distinct superimposed layers were formed, varying from 20 to 50 inches in thickness, showing that the springs were intermittent, the layers being separated by deposits of gravel cemented by lime.

Onyx-marble is also reported from the State of Oaxaca, Mexico; but little is known concerning it.

Other foreign sources are Egypt and Algiers. The Egyptian quarries are at Benisouef, about 62 miles south of Cairo, on the

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* *Engineering and Mining Journal*, vol. lvi., p. 30, July 8, 1893. The description is evidently based on a report by Prof. Merrill. An account of New Pedrara by Prof. Merrill will be found in Rothwell's *Mineral Industry*, vol. ii., p. 485.

Nile, and at Syout, 166 miles further south. The stone ranges in color from white to amber-yellow; that from Syout being paler, inclining to gray. The product of both localities is known commercially as alabaster, and is of a very different quality from the Mexican varieties. It is said to be of stalagmitic origin.

The Algerian stone from the quarries of Ain-Tembaleck, near the river Issur, is found in irregular beds from a few inches to nearly 10 feet in thickness. Its manner of occurrence has not, to my knowledge, been described; but the frequent appearance of a fibrous structure is significant.

Inferior stalagmitic marbles are quarried in many places in Italy, in the Jura mountains in France and in the vicinity of Stuttgart in Germany. The caves at Gibraltar also furnish small masses of a banded brownish stalagmite, which is cut into ornaments for the tourist trade.

From the foregoing summary it appears that the deposits furnishing the superior onyx-marble of commerce are found in regions which have been subjected to volcanic disturbance; that they are superficial deposits or vein-like inclosures, not connected in any manner with caves; that they are so frequently associated with active hot springs, or with other deposits manifestly resulting from hot springs, as to lead to a clear presumption that there must be a genetic relation between them and such springs; and, finally, that they occur associated with limestone rocks, or with rocks yielding large percentages of lime, such as diorite (usually 7 to 8 per cent. of CaO), syenite (about 4 per cent. of CaO), volcanic tufa (4 to 6 per cent. of CaO) and dolerite (often as high as 10 to 11 per cent. of CaO). It is also to be noted that the "cave-onyxes" are usually either transparent or opaque, and, so far as my experience goes, never exhibit that exquisite translucency recognized as the chief charm of the high-grade onyx-marbles which have resulted from hot-spring deposition. The "cave-onyxes" are, moreover, usually fibrous in structure, and are made up of concretionary layers, which can be scaled off like the skin of an onion. These latter peculiarities, however, are less likely to occur in the flat floor-deposits of caves, while the concretionary structure is the more common attribute of stalagmites and the fibrous structure of stalactites. The fibrous structure may occur, however, in any

situation, and is always perpendicular to the surface of deposition; and where this surface is curved, as in a stalactite or stalagmite, the fibre-like crystals extend from the center radially to the exterior; the axes passing without interruption through successive concentric layers, which may be so loosely adherent as to be split off with a light blow of the hammer.

In their other physical characters no difference seems to exist between "cave-onyxes" and hot-spring onyx-marbles. They are all calcites, as appears from their optical properties, and their specific gravities, although many writers class them as varieties of aragonite. The distinction, however, is clear, both optically and by density, none of the "cave-onyxes" or true onyx-marbles rising as high as 2.9, which is the lowest limit for the density of aragonite. The large number of specimens from caves and hot-spring deposits in all parts of the United States and Mexico which I have examined, show specific gravities ranging from 2.631 to 2.751. In composition they are exceedingly variable. The "cave-onyxes" usually contain the smallest proportion of impurities, although the floor-deposits are often rich in ferric oxide and alumina. Those from Virginia show as much as 2 per cent. of magnesia, with small amounts of manganese; and one remarkable sample yielded nearly 2 per cent. of lead sulphide, and 4.62 per cent. of antimony sulphide. A sample of green Arizona onyx-marble gave 99.84 per cent. of lime carbonate, and mere traces of iron and alumina. From 2 to 8 per cent of iron and manganese is not uncommon; but, so far, no copper or nickel has been discovered in these stones.*

The circumstances causing the great difference in texture and translucency between the "cave-onyxes" and hot-spring onyx-marbles have not yet been fully determined; and there is opportunity for trained observers to render valuable service in this particular. The greater degree of concentration of the hot solutions has been undoubtedly an important factor, and it may have been the determining one. Rapidity of flow also exerts an influence, the greater the velocity the more rapid the deposition; a circumstance first pointed out in connection with travertine deposits, I believe, by Lyell. In caves this becomes

* According to Prof. Merrill.

very conspicuous. On a sloping roof, for example, the stalactites increase in number and size towards the steeper portions where the flow of the oozing waters is greatest, and incrustations form thickest upon the vertical walls, thinning out upon the floor unless obstructions favor the building up of ledges, resulting in basins. In such cases the ledge grows upward and outward, but the incrustation again thins out upon the floor beyond, where the flow of the water is checked.

The source of the lime carbonate in cave-waters is of course the surrounding limestone, taken up by the feebly-solvent vadose circulation. They are consequently weak solutions, whereas the deep-seated plutonic waters, under high pressure and temperature, become highly charged with mineral matters. It is difficult to understand, however, that such waters rising from great depths should be so rich in lime carbonate, and yet contain so small a proportion of other ingredients, as to deposit onyx-marbles running as high as 99 per cent. in lime carbonate. The question seems a fair one, which I should like to have answered, whether the other mineral matters may not have been deposited from these solutions in the course of their ascent, and whether they may not then have derived their lime carbonate from rocks near the surface. The frequent connection of such deposits with superficial limestones and other highly calciferous rocks, tends to confirm this suspicion. That the other mineral matters should have been largely deposited below, leaving the lime carbonate still in solution, appears hardly tenable; for there is good reason to believe that mineral compounds are deposited in the inverse order of their respective heats of formation, or at least that there is an approximation to such an order, and, if this be true, lime carbonate should be deposited much earlier, and hence lower down, than a large proportion of the other substances which such waters would be expected to carry; its heat of formation being as high as 172.4.

It appears that the formation of the translucent compact variety of travertine, known as onyx-marble, is therefore due to exceedingly rapid deposition of lime carbonate from highly concentrated solutions, probably in rapid motion. Absence of pressure seems to be another requisite, judging from the circumstance that deposits of lime carbonate occurring in deep situations, as shown in metalliferous veins, take the form of

well crystallized calcite. While crystallized calcite does not require pressure for its formation, it would appear that the fine translucent variety, *i.e.*, the onyx-marble of commerce, can only be formed under normal atmospheric pressure. Further data concerning the character of the vein-like masses of onyx-marble, such as those at Antigua Salines, at a considerable distance below the surface, would be desirable as bearing upon this point. Finally, it may be indicated that, guided by empirical knowledge, prospectors would do well to search for this valuable stone in volcanic regions where hot springs do now, or formerly did, exist in close association with superficial accumulations of limestones, or limebearing plutonic and igneous rocks.

The Gold-Regions of Georgia and Alabama.

BY WILLIAM M. BREWER, ATLANTA, GA.

(Atlanta Meeting, October, 1895.)

History.—The history of gold-mining in Georgia and Alabama antedates the discovery of gold in California. A very large proportion of the gold used in the United States previous to 1849 was produced by these States. A great deal of this history is merely tradition, and no reliable statistics can be obtained; yet a personal examination of the gold-fields of Georgia and Alabama indicates that tradition is as fairly reliable, with regard to the history of these gold-regions, as it is in any similar cases throughout the world.

In Georgia, the history of gold-mining dates back to 1829, when the first gold-mining excitement in the United States occurred. In Alabama, the discovery of gold was made some years later through the work of prospectors, incited by the rich finds in the Georgia mining camps. The greatest product of gold in these States was between the years 1840 and 1849. In Lumpkin county, Ga., and Cleburne county, Ala., the districts from which the most of this gold was taken, the tailing-dumps are silent witnesses of the extensive operations which were carried on by the primitive and crude methods then known. It is no rare thing to see trees 12 to 15 inches in diameter, and even

larger, which have grown in the abandoned pits and on piles of tailings since operations were suspended. Of course, all the gold-mining carried on at that period was placer-mining, and it was not until some years later that the auriferous quartz, which occurs throughout the Appalachian chain, received any attention. Miners who were then engaged in the business have assured me that, because of the lack of knowledge of the metallurgy of gold, the first efforts at quartz-mining were really conducted on the same crude principles as the placer-mining. Such rude appliances as the arrastra, the Chilean mill, or stamp-mills constructed (except the mortars) entirely of wood, were the only ones known to these pioneers. In many instances I am told that quartz was found of such richness that a man could make wages by pounding it up in the mortar with a pestle and panning the crushed rock. These primitive methods appear to have remained in vogue to a much later time here than in the West. Indeed, I know of arrastras and wooden-stem stamp-mills which have been erected in the Southern States within the past two years, and are operated at the present time. There is some question whether the first stamp-mill of modern type was erected in Georgia or Alabama, but from what I can learn I am inclined to the opinion that McDuffie county, Ga., is entitled to the distinction of having the first modern stamp-mill operated within its boundaries. Barrel-chlorination was introduced in South Carolina at an earlier date than in western mining camps, yet the introduction of this process in Georgia only occurred in 1893; and in Alabama there is not to-day a single plant for the chlorination of sulphurets.

It is not necessary to depend on history for a knowledge of the methods of mining quartz generally adopted through these States in the past. The examination of the old openings demonstrates the fact that the miners, after discovering pay-ore in an outcrop, proceeded to take it out by following the body with a pit or inclined shaft until water interfered, or the ore became sulphuretted, or timbering became necessary. Apparently any one of these reasons was sufficient to cause the suspension of work in that particular opening. A return was then made to the surface, which was prospected along the line of strike of the formation, until another outcrop of gold-bearing

quartz was discovered, and the original operation was repeated. In the Goldville district, Tallapoosa county, Ala., as well as in the Arbacoochee district, in Cleburne county, in the same State, these methods of mining are probably more noticeable than in many other districts, because the quartz veins maintained their continuity there with greater regularity than elsewhere. In the Goldville district, the series of abandoned pits, in almost an air-line, extends nearly 4 miles. On the Anna Howe property, in the Arbacoochee district, the line of pits is nearly a mile in length; and at the Pinetucky mine, in Randolph county, Ala., the same conditions occur.

Geology.—The question, what portion of the rocks in this region belong properly to the Archæan period, has never yet been decided by geologists. Dr. Eugene A. Smith, State Geologist of Alabama, who has had more experience with this formation, at least in Alabama, than the majority of geologists, merely describes it as semi- and fully-crystalline, without giving a decided opinion as to the geological period to which it belongs. This area of semi- and fully-crystalline rocks enters Georgia from North Carolina on the north and South Carolina on the northeast, and extends across Georgia into Alabama, having Wetumpka, on the Coosa river, at its southwestern extremity. The region covered by these rocks has a triangular shape and the area is very extensive. The southern boundary, from Augusta, on the Savannah river, to Wetumpka, on the Coosa river, is about 300 miles in length; the northwestern boundary, which may be represented by a line drawn from the North Carolina-Georgia boundary, commencing in the eastern part of Fannin county, across Georgia and to the Coosa river near Wetumpka, in Alabama, is about 250 miles in length; while the extreme width from northwest to southeast is about 300 miles, as represented by a line drawn from Fannin county, Ga., to Augusta. The semi-crystalline slates occupy a much larger area in Alabama than in Georgia. It is noticeable, that while, in Georgia, the auriferous quartz is usually inclosed in the fully-crystalline mica and hornblende schists and gneiss, in Alabama it is more frequently found in the semi-crystalline or the clay-slates. In fact, in the latter State, so far as has already been discovered, I know of only one "lead," as it may be termed, of gold-bearing ore occurring in the fully-crystalline

formation, while there are five or six distinct belts or leads in the semi-crystalline and the hydro-mica slates.* In Georgia, on the contrary, nearly all the occurrences of gold-bearing ore are inclosed in the fully-crystalline schists; the only exceptions to this rule, so far as my personal observation extends, occurring in Harralson and Paulding counties, where the country-rock is a semi-crystalline slate, and in McDuffie county, where some of the country-rock is a hydro-mica schist, as well as in Gwinnett and Hall counties.

While the lines of strike and direction of dip through this entire gold-bearing region are regular, when considered as an entirety, yet locally, through faulting and folding, the formation is very complicated. The line of strike of the entire area is about N. 40° E., the dip of the strata varying from about 18° to 65° towards the east and southeast. It is often observed, however, that in local complications the trend of the formation points towards almost every point of the compass, while the direction of the dip also varies. So far as has been discovered, the northwestern portion of this crystalline region, in Georgia, contains most of the occurrences of gold-bearing ore; but in Alabama, the northwestern portion, which is composed exclusively of semi-crystalline slates, is barren, so far as prospecting has yet demonstrated, northwest of the Arbacoochee district, in Cleburne county. The expression is often heard, "I have followed this lead from Dahlonega;" but such a statement must be taken with a grain of salt, in my opinion, because my own observations lead me to say that while the occurrences of gold-bearing rock, so far as known, are arranged in certain belts, and their geographical position might lead to the supposition that they could be traced along the lines of strike from one to the other, yet when such an attempt is made, it almost invariably appears that the occurrences do not maintain continuity along their lines of strike throughout the gold-bearing area. Actual prospecting has demonstrated that the belts which contain gold-bearing ore are often entirely barren for several miles. The gold-bearing ores are really found in

* Besides the "lead" mentioned, the occurrence at Pinetucky, in Randolph county, is also in the mica-schist; and a narrow strip of gneiss, in which occur lenses of gold-bearing quartz, has been exposed quite recently in the Arbacoochee mining-district.

separate districts along the same belt of formation. For instance, a mineral map of Georgia shows a gold-belt extending from the northeastern corner of Rabun county, in a southwesterly course, to the Alabama line in the western portion of Carroll county, and another extending from the Savannah river, in Habersham county, without a break, to the Alabama line, where it crosses Troup county. As a matter of fact, the gold-bearing character of these belts cannot be traced without break; but in certain districts along the belts both auriferous alluvium and auriferous quartz have been discovered, and these districts bear such a geographical relation to each other that, when marked on the map, the gold-bearing portions appear to be connected.

Timber and Water Supply.—With regard to these important factors of mining industry, the gold-fields of Georgia and Alabama are especially favored. The timber-supply in the mountains embraced in this region is not only practically inexhaustible in quantity, but a large proportion of it, being of first growth, is very desirable for mining timbers, because it can be either sawed or hewed to 8 by 8 inches or larger, and the entire stick be all "heart." Into many of these counties, especially in Alabama, saw-mills have never penetrated, because remoteness from railroads has involved expensive wagon-hauling, while the local demand for lumber was too light to render the business profitable. In these very counties, too, prospectors are almost unknown. Consequently, their mineral resources are not only entirely undeveloped, but have not been even partially prospected.

Present Conditions.—Of the present condition of the gold-mining industry in these States, and at the different mining properties, a great deal could be written in description, did space allow for the consideration of each individual mine or prospect, or even for the mines in each county. As that is not possible in this paper, and as most of the literature on this subject to-day is descriptive of Lumpkin county and the adjacent counties in northeastern Georgia, I shall confine myself to a brief description of the conditions as they exist in Cherokee, McDuffie, Gwinnett, Hall and Carroll counties, Ga., and, because of the comparative inactivity in Alabama at the present time, shall treat of that State as a whole.

Cherokee County.—This county possesses the only really deep workings in either of the two States. At the old Franklin mine work is being carried on at an inclined depth of 507 feet, representing a vertical depth of about 450 feet. It is here that the only barrel-chlorination plant in Georgia has been erected. This was done in 1894, and since its completion the problem of saving the values contained by the sulphides in this ore has been solved both satisfactorily and profitably. A previous attempt to treat the concentrates by the cyanide process was a failure. The underground workings at the Franklin have been carried along the line of strike of the ore-body, in a northeasterly direction, under and across the Etowah river, which cuts the formation on the Franklin property and furnishes water-power for running the machinery in the stamp-mill. An examination of these workings shows that the ore-body, at some points in the mine, had attained a maximum thickness of 16 feet, but in following the line of strike, the thickness had varied from a few inches to this maximum. The values carried by the ore are stated to vary from a few dollars to $60 or $75 per ton. Recently, new pay shoots have been discovered to the southwest of the old workings, from which a great deal of this high-grade ore has been mined, and is now lying on the ground. As these discoveries are only recent, the workings are, of course, shallow, and the full extent of these shoots is not at present known.

The plant for the treatment of the ore (which, as I am informed, yields about 40 per cent. of its assay-value in free gold, with the balance carried by the concentrates) is modelled after the plant at the Haile, South Carolina, mine. It consists of a 20-stamp mill, "Embrey" and "Triumph" concentrating-tables, roasters and barrel-chlorination plant. The capacity is 50 tons per day, and the cost of milling and chlorinating from the crude ore to the bullion brick, I am assured, does not exceed 65 cents per ton.

The country-rock, which forms the walls of the ore-body, is a mica schist. The ore-body itself is a bedded vein, which was apparently formed by the folding of the formation by lateral pressure, and the mineral being deposited in crevices. Decomposition of the country-rock has extended to a depth of about 115 feet, as, I am informed, was demonstrated in the sinking of the vertical shaft a few years since. This decomposed schist

carries, or did carry, in the shallow workings, as much and often more gold than was carried by the quartz, which formed the ore-body. It is reported that the owners of the property, before the war, took out over $100,000 from this material, which was free-milling, and a great part of the value of which was saved in sluice-boxes, or "long toms."

The belt of formation in which the Franklin ore-body occurs, can be traced very readily through Cherokee county. According to the mineral maps of the State, published by the Department of Agriculture, this belt is continuous from the northeastern corner of Rabun county, in the extreme northeast of the State, to the western side of Carroll county, where it enters Alabama, and maintains its continuity across the eastern portion of that State, with the same northeastern and southwestern strike as characterizes it across the State of Georgia.

Besides the Franklin mine, the Chester, Coggins, Browley and Georgiana mines are now worked in this county. These cannot be considered producers, and the work upon them is at present merely of a prospecting nature. The industry in the county employs about 200 men. Nearly all the mines I have referred to, as well as the Worley and Kellogg, have been sunk through the decomposed formation, and work has been suspended, either at that depth or within a few feet beyond the line of demarcation between the decomposed and the solid formation.

Carroll County.—The gold-bearing ores in this county occur as I have said, in the same belt of formation as those in Cherokee county. But, so far as discoveries have determined, there are several breaks in the continuity of the auriferous bodies of quartz along the line of strike; so that while the formation maintains its continuity, there cannot be said to be a clearly defined "lead" of auriferous quartz, as that term is usually understood in mining parlance. Considerable activity is being manifested at present in the vicinity of Villa Rica, which was one of the early gold-mining camps. From the evidence of the tailing-dumps, and the extensive open cuts, it would appear that this section had been one of the most extensively worked in either of the States. An English company, which has recently bonded mining property near here, purposes sinking 300 feet to ascertain the character and extent of the ore-bodies at that depth. Considerable interest is manifested as to the results,

which will be shown from this work, and also from the work being done by a Boston syndicate on the property known as the old Clopton mine, which was one of the earliest and most extensively worked gold-placers in the county. On this property an entirely new and (at least in this section) untried process for the treatment of sulphuretted ore is to be adopted. The plant is almost completed, and before this meeting closes, the probabilities are that the process will either be established as a success, or will have proved itself a failure. According to the inventor's description, this process is apparently a combination, consisting in amalgamation aided by electrolysis.

Immediately adjoining the Clopton mine on the south and west, is a property which has been in active operation on a small scale for some years past. It is worked very economically; and as the sulphuret-ore has not yet been reached, the value is saved by amalgamation. An examination of the open cut, which has been the method of mining, to a depth of about 30 feet, shows that the decomposed mica schist, as well as the lenses of auriferous quartz, carried value for a thickness of about 20 feet. In the floor of this open cut, a shaft has been sunk some 30 feet, and a drift has been run on that level. While this work has not been sufficient to demonstrate many facts regarding the ore-body, it shows that the ore, as it nears water-level, is becoming sulphuretted, and apparently the body itself is becoming more compact and concentrated. But a sufficient depth has not been attained to expose rock in place.

As an old miner once expressed himself to me: "This country is decomposed to such a depth that it is in but few places that the outcrop of the ore, as we consider it in the West, has been discovered." He meant that in the West this immense superincumbent mass of decomposed material being generally wanting, the solid formation would be exposed as the surface-outcrop. After a close study of the conditions of gold-mining in Georgia and Alabama, I am convinced that, because of the formation of the ore-bodies, while operations on a limited scale, guided by experience, judgment and conservatism, will undoubtedly, in many instances, result profitably, as is the fact in this particular instance, yet, at the same time, operations on a very extensive scale are more likely to result disastrously, unless the property is thoroughly and systematically developed before extensive machinery is introduced.

Gwinnett County.—In this county, near Buford, where the Piedmont mine is being developed, to ascertain what method of treatment and what scale of operations will be advisable, I find a well-defined true fissure-vein. The dip is vertical and the strike a few degrees north of west, while the dip of the country is about 20 degrees east, and the line of strike northeast and southwest. Near the surface, this vein is almost imperceptible, but at a depth of 85 feet, it has increased in thickness to nearly 3 feet. The country-rock is gneiss. The ore is a hard white quartz, occasionally showing native gold, and a large proportion of it carrying a heavy percentage of iron pyrites and argentiferous galena. A drift, 185 feet long, shows that the vein maintains its continuity, at least for that distance. Among the other occurrences of gold-bearing ore in this county, I fail to find another well-defined fissure-vein. But I am not prepared to say that such do not exist, because there has been, in the past, work of considerable extent performed on several properties within a short distance of the Piedmont. There are also evidences that the Mexican arrastra was used to treat the ore from these workings. From the size of the trees which have grown on the tailing-piles, and in the pits, it is evident that this work was performed nearly, if not quite, a half-century ago. All the ore from Gwinnett county, even above water-level, carries a larger percentage of galena, than in any other county I have observed in the State, with the exception of McDuffie and Wilkes. There appears to have been less placer-mining in this county than is usually the case where the occurrences of gold-bearing ore-bodies occur in as many localities as in this county. From what history I could gather, and from the best opinion one could form from the workings, Gwinnett has always been a quartz-mining, rather than a placer-mining region.

Hall County.—Near Gainesville, in this county, active mining is going on at three places, namely, the Merck mine, a mile and a quarter from Gainesville; the Currahee, near White Sulphur, 6 miles northeast of Gainesville; and the Potosi, about 12 miles in a northwesterly direction from Gainesville. The first two of these occur in a hydro-mica schist country-rock, and the last in the gneiss. The Merck is a recent discovery; and two miners from Colorado have bonded the property, and are now pros-

pecting. The work has not progressed sufficiently to expose rock in place. In a tunnel which cross-cuts the ore-body about 30 feet below the surface, it shows a thickness of about 5 feet, but is very much broken up; and until the work is prosecuted further, no reliable opinion can be formed of its characteristics. The ore is very heavily sulphuretted and carries a considerable percentage of galena, even close to the surface. There is some reason for the opinion that further development may demonstrate this ore-body to possess the characteristics of a true fissure-vein, in that it appears to cut the formation by dipping towards the north while the formation dips towards the south-east; but the strike is conformable with that of the formation. It is purposed by the prospectors of this property, if the ore-body maintains its continuity, to erect a plant for the treatment of the ore which in its construction presents decidedly new features, as compared with any of our southern plants, but, as I am assured, is producing very satisfactory results in the vicinity of Ouray, Col. In this process (known as that of the McCoy patent) a series of hopper-shaped vessels, designed to take the place of the usual amalgamating-plates, as well as to aid in mechanical concentration, receive the pulp from the stamp-battery. Compressed air is introduced into these vessels to agitate the pulp and to bring (as is claimed) every particle of free gold into contact with the mercurialized plates with which the vessels are lined. It is asserted that in this way all loss in slimes is prevented, because the air forces every particle of the pulp to the bottom of the vessel, preventing the removal of valuable material in the slimes, which so often occurs in concentration as well as in amalgamation.

At the Currahee mine, five prospecting-tunnels have been run, the longest of which is 460 feet, and in the face a body of ore is exposed which is clearly a bedded vein, having strike and dip conformable with the enclosing formation. The ore is extremely refractory, carrying a large percentage of iron pyrites, galena, zinc and arsenic, as well as some antimony. In order to overcome the rebellious nature of this ore, the company purposes testing a method of treatment, which is described as "dry chlorination," and which has been practiced in the laboratory, as I am informed, with satisfactory success, but to the present time, has not been proved on a commercial scale. The

last-named of these locations, the Potosi, is one of the oldest placer-mines in Georgia, having been worked as early as 1830. The decomposed gneiss formation, with lenses of quartz bedded within it, has produced some of the richest specimens of native gold found in the State. Several specimens of quartz with native gold, have been taken from the property, which would assay not less than $40,000 per ton. This property is now being prospected for auriferous quartz. A prospect-shaft 80 feet in depth had been sunk at the time of my recent visit, from the bottom of which a cross-cut had been run, and apparently a lense of quartz had been cut. Although it proved barren, a drift was started and run some 30 or 40 feet on it, from the fact that at the surface a very rich shoot of ore had been worked down, to a shallow depth. It was the intention of the operators to continue this drift until it had underrun a very rich shoot of ore formerly worked on the surface, and in this way to determine the nature of that shoot at the deeper level.

Dredge-Boats.—Several years ago, dredge-boats are said to have been profitably worked on the Chestatee river. In 1894, Mr. Jaquish, of Gainesville, tried the experiment of operating such machinery at Louisville. He has kept his boat in continuous operation since then, and there have been two others put in the river at different points above him. The sand and gravel from the bed of the stream is hoisted into sluice-boxes arranged on the boat, and the gold is saved by the same method of sluicing as in placer-mining. It is impossible to ascertain with any degree of accuracy, the quantity of gold which has been saved on the dredge-boat within the past year. But I have every reason to believe that the aggregate is large. The fact that two other boats have been put in operation since Mr. Jaquish commenced, is certainly a very encouraging sign. The Chestatee river drains Lumpkin and White counties, which, according to history and statistics, have been the steadiest producers, and have produced the largest amounts in nuggets and gold-dust of all counties in the State. The operations conducted on this river alone, including hydraulicking, dredging and sluicing, are of sufficient importance to be described in a separate paper.

McDuffie and Wilkes Counties.—Of this section of the gold-

bearing region, little has been written. Indeed, even in our own State, the statement is generally received with surprise, that the most successful quartz-mining in the State is being done in this district. Such, however, is the fact. It is natural to consider these two counties together, because McDuffie is comparatively a new county, and was originally cut off from Wilkes. Quartz-mining and treatment of the ore by amalgation in a five-stamp mill, furnished with 450-pound stamps, and run by the water-power supplied from Little river, have been carried on for upwards of thirty years, almost continuously. Col. J. Belknap Smith, I am inclined to believe, was the pioneer in successfully treating auriferous quartz by amalgamation in the State of Georgia. To-day, his widow is certainly the most successful gold-miner in the State. During my recent visit, she demonstrated to me that, with only five stamps dropping, and crushing not to exceed six or seven tons of ore a day, and while hauling the ore three miles and a half from the mine to the mill, she is doing a profitable business. The value of the ore varies from seven or eight dollars a ton, up to very high grades, Mrs. Smith herself having extracted by amalgamation as much as sixty dollars per ton. She has mastered the rudiments of mining and amalgamation sufficiently to superintend operations personally both at the mine and the mill. The workings in the mine may be described as of two sets, the old and the new, or, in other words, the primitive and the modern systems of mining. By the first of these, the ore was taken out to an inclined depth of some 200 feet from the outcrop; and to-day the old pits, which have caved in and are partially filled with *débris*, from which is growing a grove of large pine trees, demonstrate how extensively work was carried on before the war.

The workings by the modern method, which are at present in active operation, demonstrate the systematic and practical business sense of the management. I was unable to explore any of the old workings, because of their condition; but I was cordially invited to examine the workings in the mine as they are at present conducted. I found two vertical shafts, each 160 feet deep. From the first one, known as the Wilsher, the ore has been stoped out on several levels to the depth of 120 feet, except immediately back of the shaft, where a pillar of solid ore,

30 feet long, averaging 4 feet in thickness, and from the 50-foot level to the 160, has been left intact as a support for the shaft. Samples taken on the various levels, from this body of ore, show that it is of high grade. On the 160-foot level, from which the miners are now stoping, the ore in the drift has become more sulphuretted, but still retains much of its free-milling nature, although it is over 100 feet below water-level. In fact, between $20 and $30 per ton are saved by amalgamation in the mill. The ore-body in this mine presents all the appearance of being a true fissure-vein, cutting the formation diagonally, with its line of strike almost north and south, while the line of strike of the formation is northeast and southwest. The walls are well-defined, of hydro-mica slate. The vein itself is hard white quartz, occcasionally carrying a percentage of galena and iron pyrites, and frequently showing very beautiful specimens of native gold. Immediately adjoining the vein, on the foot-wall side, is a blue quartz, heavily charged with cubes of iron pyrite. Its value, if it has any, is unknown, because Mrs. Smith never has had any assays made, and ore that will not pay by amalgamation, is sent to the waste-pile. No apparatus for concentration is attached to the mill; and consequently, for the past forty years, whenever sulphuretted ore has been milled, a large proportion of its value has passed into the river. Occasionally, when the river is low, it has been the custom to work this bed of tailings through sluice-boxes, the result always being that the miners more than earned their wages. Of course, the question arises, Why are not these concentrates saved by machinery, and suitably treated? To this the proprietress replies, that considering the cost of the extra machinery, she doubts whether the yield from the concentrates would have been sufficient up to this time to repay the first cost of machinery. But she readily realizes that some such method will have to be adopted in the future if she continues to increase the depth of the mine-workings.

In the second or Smith shaft a vertical depth of 150 feet has likewise been obtained. It was purposed to intersect with this shaft, or near the bottom of it, an ore-body supposed, from the outcroppings, to lie parallel with that being worked in the Wilsher shaft. But the work has demonstrated that this theory

was erroneous. Instead of a separate parallel ore-body, a narrow streak of auriferous quartz was found in the drift run from the bottom of the Smith shaft. Its strike was directly towards the main ore-body, which it will apparently intersect within 50 feet of the present face of the drift. In thickness this quartz averages only a few inches, but in value it mills about $30 per ton. No levels have been opened up from this shaft and no stoping has been done; consequently, it is impossible to express any opinion regarding the permanency or character of the ore-body, or what relationship it will be found to bear to the main vein. One fact is demonstrated, namely, that this vein is bedded conformably with the dip and strike of the formation, and has lenticular structure. While this mine has received special notice, because of the long period through which it has been a regular producer of bullion, yet, from my examination, I am satisfied that, although there may possibly not be another fissure-vein in the district, or even an extension of Mrs. Smith's, yet there are several prospects in which both grade and quantity of ore are sufficiently determined to warrant the expenditure of capital in development. The prevailing country-rock in the district is a hydro-mica schist, in which the gold is usually carried, but there are also occurrences of gold in the fully-crystalline mica, gneiss and hornblende schists.

Alabama.—The various belts which occur in Georgia cross into Alabama, and some of them extend upwards of fifty miles beyond the Georgia line. The gold-bearing area in Alabama, embracing only the counties of Cleburne, Randolph, Clay, Talladega, Tallapoosa and a portion of Coosa, is much smaller than in Georgia. Gold-mining in Alabama is not as active as it was in the spring of 1893. In fact, during the present year the only activity that amounted to anything has been shown in Cleburne county, in the old Arbacoochee district. The traditions of this district are legion. Several survivors of the days of Arbacoochee's greatness remain to boast of the good old times when, as they assert, 1500 to 1800 men were regularly employed in the placer-mines, and the camp, at night and on Sundays, was filled with the roystering, boisterous society characteristic of a booming mining camp. The work done during the present year has been chiefly that of reopening and unwatering

some old pits which were sunk about 40 years ago, and from which, as tradition said, the miners were driven out by the excessive influx of water. The sons of an old miner have insisted for a long time that they could locate a very rich body of ore on Section 7 and a portion of Section 6, known as the Denson property, which lies adjacent to the Arbacoochee property. It was not until last spring, however, that they obtained a bond for title, and commenced prospecting-work. Within a few months after they demonstrated, by the production of very high-grade ore, the truth of their claim that they had obtained reliable information directly from one of the original miners.

Operations have been resumed during the present year, at the old Pinetucky mine in Randolph county, which was among the oldest quartz-discoveries in the Southern States. Sinking was continued on a shaft, which was started in the mill-house, and which, if carried down to a vertical depth of about 150 feet, ought to intersect the ore-body on its dip and open up for mining and stoping about 200 feet of virgin ground. The ore-body in this Pinetucky mine averages about 10 inches in thickness in the drift of the deepest level of the old workings, which is 55 feet vertically below the surface. It is a very hard blue quartz, carrying a large percentage of sulphides and really of such a refractory nature below water-level that, if operations are continued, the same methods as have been adopted at the Franklin mines in Georgia will be necessary here. Much of the ore is shown by assay to be of a very high grade, often yielding from $150 to $200 gold per ton. The pay-ore lies in shoots very similar to those of the Franklin mine. This has been demonstrated by drifting on the 55-foot level of the old workings. A modern Frazer & Chalmers mill of ten 750-pound stamps was erected on this property in 1890, and the mine has been worked since then at irregular intervals by lessees. The history of this mine reaches back to 1850, when the decomposed mica-schist country-rock and the gravel in neighboring streams were found to yield gold by panning. A further search revealed the occurrence of auriferous quartz sufficiently oxidized to yield its values to amalgamation; and one of the crude, small stamp-mills, called "pounding-mills" in those days, was erected on the property, and for several years was run continuously on the oxidized ore found above water-level. For

more than a mile and a half, the surface has been mined in a series of pits extending almost in an air-line, throughout that distance, with an average depth of some 30 feet. In this surface-mining, as in the underground workings, the distribution of the pay-ore in shoots has been demonstrated. It is reported that as much as 800 dwts. of crude gold have been cleaned off the plates from one battery, as the result of one afternoon's run on this surface-ore. Towards the southwestern extremity of the Alabama gold-field, and especially in Tallapoosa county, the gold-bearing quartz occurs in a country-rock of graphitic slate, in such a way that the ore itself cannot be mined profitably unless graphite is taken with it. In treating such ore by amalgamation, the quicksilver is "sickened" as soon as the graphite comes in contact with it, in passing over the plate, and the amalgam, when cleaned up, is found to carry a large percentage of graphite with the gold. In the Silver Hill district, Tallapoosa county, where Major Parmelee, of New York, has been operating mines for several years past, this difficulty presented itself in the treatment of the surface or oxidized ores, and has not yet been thoroughly overcome, although the superintendent has made a special study of the matter. It is in this district, that the graphitic slate is associated in much larger quantities with the auriferous quartz, than in any other part of Alabama or Georgia. At one hill, known as Blue Hill, and composed of graphitic slate, with interlaminated lenses of auriferous quartz, the close association of the slate and quartz, the narrow and irregular character of the quartz lenses and the fact that the graphitic slate itself is auriferous, render any separation at the mine both difficult and inadvisable, consequently the hillside has been quarried off for a total width of some 50 or 60 feet; and, at the time I saw it, the face of the excavation measured nearly 100 feet from the floor to the surface.

Another interesting district in Alabama is the "Hog Mountain," also in Tallapoosa county, where one body of auriferous quartz is 30 feet thick, and several others range in thickness from 6 feet upward. No very extensive work has ever been conducted on the property, although it has been prospected by shallow openings at several points, and a small millway run on the ore several years ago. Litigation among the owners is, as I am informed, the reason that operations were suspended.

Apparently, the supply of ore, provided all the bodies of quartz carry value, is inexhaustible. But, like too many other properties in this southern country, the work of prospecting was suspended before facts sufficient to disclose the value of the property had been determined. In this mountain, gold-bearing quartz is inclosed in the fully-crystalline schists, and also in the semi-crystalline slates, the mountain itself being made up of mica schist and gneiss, while the valleys at its base are in semi-crystalline slate.

The present inactivity of gold-mining in Alabama is, to a great extent, the result of some failures made in 1893 in the northern portion of the gold-field. These were largely attributable to the fact that the operators erected plants for treatment before they had ascertained the quantity, character, and value of the ore they proposed to treat. Another cause for the present inactivity is the circumstance that many properties are in litigation. Another is that the owners of properties, which are suspected of containing gold-bearing rock, rate the mere chance of mineral value at so high a figure, and consequently hold their undeveloped lands at such extravagant prices, that practical mining men cannot afford to take the chances of purchasing.

The most important districts in Alabama are the Turkey Heaven (named from the mountain of that name), Arbacoochee, Chulafinne, Riddle's Mills, Idaho, Kemp Mountain, Hog Mountain, Goldville, Crooked Creek, Gold Ridge, and Silver Hill. These districts are located on five different gold-bearing belts, which are separated from each other by intervals of several miles. The ore-bodies in all of them are bedded conformably with the formation, and are usually of lenticular form. Below water-level all the ore loses its free-milling character and becomes sulphuretted; but in some districts, especially the Crooked Creek, the ore is much more refractory than in others. Indeed, at many places in this district, on the extreme western boundary of Randolph and eastern boundary of Clay counties, the ore is refractory from the grass-roots down. A large proportion of the gold-bearing ore in that section is mispickel. In view of this circumstance, and also of the absence of any plants in the district for the treatment of the ore by a hydro-metallur-

gical method, it is not surprising that but little activity is manifested in the industry. Prospecting, however, has developed the fact that, as compared with other districts in the gold-bearing region, the ore-bodies here are apparently of much greater extent. Developments have undoubtedly been discouraged by the distance (nearly 40 miles) from the nearest railroad station, and by the fact that no very important discoveries of placer-mines or of free-milling auriferous quartz have been made in the district.

The Alabama districts which have produced gold in the past are nearly all situated immediately adjacent to, or within a short distance from, either the Big or Little Tallapoosa rivers. The exceptions are the Riddle's Mills and Idaho districts. These are located, the former in Talladega county, on the western flank of the Talladega mountain, and the latter in Clay county, on the eastern flank of the same mountain.

The Rappatoe property, in Elmore county, on the Coosa river, exhibits the most southwestern occurrence of gold-bearing gravel in the State. No bodies of auriferous quartz have been discovered there, and the gravel was worked out by sluicing several years ago, although occasionally spots are found in it, even to-day, that will pay wages by sluicing.

In the Kemp Mountain district the Eckles property has been opened to a greater depth than any other in the State. A vertical shaft has been sunk 130 feet, and water-level has not yet been reached. The ore, which is in seams of decomposed quartz, interstratified with the mica schist country-rock, is comparatively free-milling, the only difficulty experienced being due to the graphite associated with the ore. The effort of the owner of this property has been to develop and show ore in sight, rather than to obtain returns from treatment. Consequently, no metallurgical plant has ever been erected. At the first discovery of this mine, the surface "prospected" for a thickness of about 40 feet, the value being carried by both the mica schist country-rock and lenses of quartz; but at a depth of 50 feet the vein-matter was only 18 feet in thickness. This thickness has been maintained as depth has been increased in the workings.

It is a noticeable fact throughout Alabama that a great deal

of the country-rock inclosing the ore-bodies carries values often greater than those carried by the quartz itself.

In comparison with western mining, the work done in this State is so shallow, and of such limited extent, that we really cannot claim anything beyond prospects, the value of which future development alone can prove. Even estimates of ore in sight are, in most cases, necessarily unreliable. In a word, the gold-fields of Alabama have only been partially prospected.

The Effect of Washing with Water Upon the Silver Chloride in Roasted Ore.

BY WILLARD S. MORSE, PRESCOTT, ARIZONA.

(Atlanta Meeting, October, 1895.)

IN my paper on "The Lixiviation of Silver-Ores by the Russell Process at Aspen, Colorado" (page 137 of the present volume), attention was called to the decrease in "chlorination" during the washing of the roasted ore with water, to remove soluble salts, before leaching with hyposulphite and Russell solutions.

This decrease in "chlorination," which is due to conversion of the silver chloride to some other form not soluble in hyposulphite solutions, and which, for the sake of brevity, will be termed the "going-back" of chlorination, has been a serious obstacle to the successful operation of several leaching-works, and was at Aspen the only difficulty encountered.

I have never seen any explanation of the cause of this failure to extract, in actual practice in the mill, the amount of silver chloride shown by laboratory-leachings to be present in the roasted ore; and the purpose of this paper is to record the conclusions arrived at concerning the cause of the phenomenon, and the data upon which they are based.

The average decrease in chlorination by washing with water in the Aspen ores for a period of fourteen months was as follows:

	Per cent.
Silver soluble in a solution of hyposulphite of soda, in the roasted ore, when delivered to leaching-vats, . . .	78.93
Silver similarly soluble after washing with water for about twelve hours, ...	64.33
Decrease in chlorination, or silver converted from chloride to some other form, not soluble in solution of hyposulphite of soda, by the process of washing with water, . . .	14.60

The above calculations are based on results obtained in the treatment of over 30,000 tons of ore, and are computed from the average of several hundred determinations.

Mr. L. D. Godshall, at one time superintendent of the Aspen works, in a "Review of the Russell Process," * advances the theory that the "going-back" of chlorination is caused by the presence, in the roasted ore, of a reducing agent in the form of sulphurous acid, reducing the silver chloride to metallic silver, insoluble in hyposulphite of soda solutions, and but very slowly soluble in Russell solutions. While superintendent of the works, Mr. Godshall could devise no method to overcome the difficulty, and, in fact, beyond the simple assertion of his opinion, could advance no proof to substantiate his theory.

In the opinion of the writer, the cause of the "going-back" of chlorination is principally the presence in the roasted ore of sulphides of the base metals which have escaped oxidation in roasting; and, in the case of the Aspen ores, this result was due to the zinc sulphide, which, during the rapid roasting in the Stetefeldt furnace, did not have sufficient time to decompose. It is well known that zinc-blende requires more time and heat for complete oxidation than iron pyrites; and I do not think that in the short time that the ore is subjected to the heat in a Stetefeldt furnace, it is possible to decompose any appreciable percentage of zinc-blende present. No difficulty was experienced in roasting in the Stetefeldt furnace charges of ore containing from 16 to 18 per cent. of iron pyrites (8 to 9 per cent. of sulphur), and completely oxidizing the sulphur; but the table given in this paper shows that while the raw ore averaged only 1.98 per cent. of zinc as sulphide (or 2.95 per cent. of blende), the roasted ore contained 1.56 per cent. of zinc as sulphide; that is, only about 21 per cent. of the zinc-blende had been decomposed in the roasting in the Stetefeldt furnace.

* *Proc. Colo. Sci. Soc.*, vol. iv., p. 306.

The fact that the final apparent extraction of silver in the roasted ore was, by the use of the Russell solution, brought up to 86.74 per cent., or 7.81 per cent. more than the amount shown to have been present in the roasted ore as chloride, proved conclusively that, whatever form the chloride had been converted into during the process of washing, it was soluble in the cuprous hyposulphite solutions of the Russell process. This led the writer to the conclusion that the silver chloride had been converted to a sulphide, which is readily soluble in the Russell solution. It was found possible, in mill-practice, to extract the silver with the Russell solutions very nearly up to the highest laboratory-results.

The experiments of Malaguti and Durocher, recorded in Percy's *Metallurgy*, showing the reactions between chloride of silver and the sulphides of the base metals, furnished a clue to the cause of the difficulty; and undoubtedly the reason for the "going-back" of chlorination, in the case of roasted ores containing zinc sulphide, is a double decomposition taking place in the brine solution made by the wash-water and the excess of salt in the roasted ore, expressed by the equation

$$2AgCl + ZnS = Ag_2S + ZnCl_2.$$

The Table on the following page shows, I think, beyond doubt, that in the case of the Aspen ores, at least, the zinc sulphide in the roasted ore was responsible for the "going back" of chlorination.

The determinations given were made by Mr. Stuart Croasdale. In the case of the raw ore the results given are the average of over one thousand separate analyses, made on each lot of ore comprising the run, while the determinations on the roasted ore were made in duplicate on composite samples made up from each charge of about 60 tons. The chlorination-tests given are the averages of the tests made on each charge. A few slight discrepancies will be noticed in the table, where the zinc present as sulphide in the roasted ore exceeds that reported in the raw ore. These contradictions can only be accounted for by irregularities in the samples.

A number of experiments made in the laboratory by Mr. C. A. Hoyt, who allowed samples of roasted ore to stand in water in presence of an artificial zinc sulphide, showed conclusively

TABLE I.—*Showing the Percentage of Zinc as Sulphide in the Raw and in the Roasted Ore and the Percentage of Decrease in Silver Chloride by Washing the Roasted Ore with Water.*

Mixture-Number.	Raw Ore in Mixture.	Zinc in Raw Ore.	Zinc as Sulphide in Raw Ore.	Zinc as Sulphide in Roasted Ore.	Proportion of Total Silver in Roasted Ore Soluble in Solution of Hyposulphite of Soda.		Decrease of Silver Chloride by Washing.
					Before Washing.	After Washing.	
	Tons.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
1	1309	2.80	2.38	2.00	83.40	61.10	22.30
2	1231	4.20	3.47	2.59	63.83	38.90	24.93
3	2090	4.90	4.34	2.86	75.91	43.50	32.41
4	1319	4.50	3.74	3.16	71.16	34.00	37.16
5	1276	3.40	3.02	2.04	80.04	63.90	16.14
6	2435	3.20	2.96	2.07	76.62	57.04	19.58
7	3255	3.70	2.86	2.18	76.25	57.46	18.79
8	2123	2.10	1.56	0.82	81.80	73.20	8.60
9	2687	2.10	1.54	1.00	84.85	81.53	3.32
10	2656	1.55	0.86	1.15	81.96	81.41	0.55
11	2860	1.80	1.21	0.68	82.33	79.15	3.18
12	2602	2.80	1.56	1.64	79.72	70.06	9.66
13	2380	2.14	1.65	1.14	79.03	69.85	9.18
14	2033	1.58	0.99	0.97	83.36	83.12	0.24
15	541	1.73	1.14	1.05	73.21	68.68	4.53
16	961	2.68	2.06	1.87	70.60	55.60	15.00
17	410	0.90	0.84	0.89	89.70	85.70	4.00
18	1065	1.81	1.31	1.25	74.31	61.72	12.59
19	1911	1.92	1.60	1.36	79.85	71.99	7.86

TABLE II.—*Results in Table I. Arranged in the Order of the Percentage of Decrease in Chlorination by Washing.*

Mixture-Number.	Decrease in Chlorination by Washing.	Zinc as Sulphide in Roasted Ore.
	Per cent.	Per cent.
4	37.16	3.16
3	32.41	2.86
2	24.93	2.59
1	22.30	2.00
6	19.58	2.07
7	18.79	2.18
5	16.14	2.04
16	15.00	1.87
18	12.59	1.25
12	9.66	1.64
13	9.18	1.14
8	8.60	0.82
19	7.86	1.36
15	4.53	1.05
17	4.00	0.89
9	3.32	1.00
11	3.18	0.68
10	0.55	1.15
14	0.24	0.97

that the silver chloride was decomposed exactly in proportion to the amount of zinc sulphide present.

In March of the present year a very complete laboratory-experiment was made for Mr. C. A. Stetefeldt by Mr. Frank A. Bird, of Park City, Utah, to ascertain whether the reaction occurred between chloride of silver and natural zinc-blende, and thus to confirm or disprove my contention, that the unoxidized zinc sulphide left in the Aspen ores after roasting in a Stetefeldt furnace, was responsible for the "going back" of chlorination.

The result of Mr. Bird's experiment shows conclusively that such a reaction occurs, and that it is more pronounced in a brine solution than in a solution of hyposulphite of soda. This would confirm my theory, as the ore roasted with 10 to 12 per cent. of salt contains an excess of sodium chloride, and the wash-water becomes, practically, a brine solution.

The following is Mr. Bird's report:

Reactions Between Zinc-Blende and Silver Chloride.

$$2AgCl + ZnS = Ag_2S + ZnCl_2.$$

Two samples of zinc-blende were used: One of crude ore, the other partially roasted. Crude ore was first ground on an iron plate to pass a 60-mesh screen, and a portion was then roasted 22 minutes, with frequent stirring, in a muffle kept at red heat. After this partial roast, the portion was reground in a porcelain mortar to pass a 60-mesh screen.

Analyses of the crude and the partially-roasted ore gave the following results:

	Crude. Per cent.	Partially Roasted. Per cent.
Zinc,	55.20	58.20
Sulphur,	37.17	16.47
Lead,	6.00	4.70
	Ozs. per Ton.	Ozs. per Ton.
Silver,	4.4	5.6
	Or 2.2 milligrammes in 0.5 A. T.	2.8 milligrammes in 0.5 A. T.

Six samples of 0.5 A. T. of each class were taken; to each of three of them was added 100 c.c. of a concentrated solution of sodium chloride containing silver (Solution "A," below), and to each of the other three was added 100 c.c. of a concentrated solution of sodium hyposulphite ($Na_2S_2O_3$) containing silver (Solution "B," below).

After the samples had stood for the time shown in Table III., they were filtered and washed until no reaction was obtained for silver, and a hot 20-per-cent. solution of sodium chloride was used to wash the samples that had stood in Solution "A," and cold water to wash those that had stood in Solution "B."

TABLE III.—*Showing Results of Foregoing Experiments.*

TIME IN WHICH SAMPLES STOOD IN SOLUTIONS.	BRINE SOLUTION, CONTAINING 64.1 MILLIGRAMMES OF SILVER IN 100 C.C.						HYPOSULPHITE SOLUTION, CONTAINING 100 MILLIGRAMMES OF SILVER IN 100 C.C.					
	Crude.			Roasted.			Crude.			Roasted.		
	Assay.	Precipitated.		Assay.	Precipitated.		Assay.	Precipitated.		Assay.	Precipitated.	
Hours.	Mgms.	Mgms.	Per cent.	Mgms.	Mgms.	Per cent.	Mgms.	Mgms.	Per cent.	Mgms.	Mgms.	Per cent.
6	44.78	42.58	66.43	13.42	10.62	16.57	40.42	38.22	38.22	11.1	8.3	8.3
12	40.64	38.44	59.97	17.60	14.80	23.09	45.00	42.80	42.80	12.38	9.58	9.58
24	45.40	43.20	67.39	17.20	14.40	22.46	51.78	49.58	49.58	12.88	10.08	10.08

After washing, the residues were assayed, with the results given under the heading "Assay," in Table III., in milligrammes of silver. From the total silver found in the residue the amount contained in the original sample is deducted, and the remainder, being the amount precipitated from the solutions, is given under the heading "Precipitated." The percentage thus precipitated was calculated upon the total silver in the solution in which the sample had stood.

Solution A. Sodium chloride (brine) of 22.6° B. *at* 60° F.—A saturated solution, made with pure water and commercial sodium chloride, was filtered, heated to boiling, and again passed through a filter upon which was 1 gramme of silver, freshly precipitated as chloride. After passing it through the filter once, to loosen all particles of silver and wash it down into the filter-point, the silver was washed with hot brine into the salt solution, and boiled until it was all in solution, and then the solution was made up with hot brine to exactly 1 liter. Upon cooling to the normal temperature, some silver chloride precipitated out, and as it was found impossible to make a stable compound, the solution was finally tested for silver, and found to contain 64.1 milligrammes per 100 c.c.

Solution B. Sodium hyposulphite of 1.5° B. *at* 60° F.—20 grammes of commercial sodium hyposulphite were dissolved in pure cold water, and passed repeatedly through a filter, upon which was 1 gramme of c. p. silver, freshly precipitated as chloride, until it was all in solution. The solution was then diluted exactly to 1 liter, so that each 100 c.c. contained 100 milligrammes of silver.

Note.—Before either solution was run over the silver chloride, the latter was washed until no reaction was given with silver nitrate.

All figures are based upon the commercial assay.

Mr. Godshall, in his paper cited above, assumes that the effect produced by zinc sulphide in the roasted ore is due to the production, by the oxidation of the zinc sulphide on the cooling floor, of sulphurous acid, which reduces the silver chloride to metallic silver, and that this metallic silver is afterwards extracted by the Russell solutions. I question the correctness of this theory; since metallic silver is, at best, only

TABLE IV.—*Excess of Extraction by Russell Solutions.*

Mixture-Number.	Highest Laboratory-Extraction by Hyposulphite Solutions.		Decrease in Chlorination by Washing.	Final Extraction in Mill by Russell Solutions.	Extraction by Russell Solutions over Best Extraction by Hyposulphite.
	Roasted Ore.	Washed Ore.			
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
4	71.16	34.00	37.16	80.8	9.64
11	82.33	79.15	3.18	90.5	8.17

slowly soluble, while sulphide of silver is readily soluble in the Russell solutions. In every case the final extraction of silver in the mill by the Russell solution was higher than the best ex-

traction in the laboratory by hypo-solutions, showing that the Russell solution not only extracted the silver that had been changed from a chloride by washing, but also an appreciable percentage that had never been converted to chloride in roasting, and presumably remained in the ore as a sulphide of silver. The examples given in Table IV. will illustrate this statement.

In connection with the subject of the mutual decomposition of the sulphides of the base metals and chloride of silver, the result of turning this reaction to practical account will be of interest. During the operation of washing with water to remove the soluble salts, about 14 per cent. of the silver was removed from the roasted ore by reason of the excess of sodium chloride present in it, silver chlorides being dissolved in the brine thus formed. The product from precipitation of the wash-water with sodium sulphide was very low in silver (600 to 1000 ounces of silver to the ton); on account of the presence of lead, which was also precipitated as a sulphide by the sodium sulphide.

The reaction, $2AgCl + PbS = Ag_2S + PbCl_2$ was utilized with splendid results. After a few lots of wash-water had been precipitated with sodium sulphide, and a stock of precipitate, consisting mostly of lead sulphide had been thus obtained, the wash-water from the ore was run into the tanks, and the accumulated lead sulphides were thoroughly mixed with the wash-water by agitating with compressed air for about two hours. The result was a complete precipitation of the silver chloride in the wash-water, and the stock of precipitates first made was constantly increased in grade of silver. The saving effected was not only in the decreased cost of sodium sulphide for precipitation, but also in the decreased cost of refining the product.

In conclusion, I believe that the reactions occurring between chloride of silver and the sulphides of the base metals are of great importance in the leaching of silver-ores, either with brine, hyposulphite of soda or the Russell solutions; and, from all the information I can gather, this point seems to have been overlooked by those who have investigated the subject, or at least by those who have published the results of their investigations.

The Geological Structure of the Western Part of the Vermilion Range, Minnesota.

BY HENRY LLOYD SMYTH, CAMBRIDGE, MASS., AND J. RALPH FINLAY, VIRGINIA, MINN.

(Atlanta Meeting, October, 1895.)

I.—INTRODUCTION.

THE most important area of the so-called Keewatin rocks of northern Minnesota is that which runs westerly from Lake Saganaga, near the national boundary, and finally disappears beneath the drift (or has not been farther traced), in the neighborhood of Vermilion Lake. With this belt we have a general acquaintance, extending back some four years. During the past year we have had the opportunity of studying in considerable detail a small part of it, embracing an area of about 12 square miles on the south and east shores of Vermilion Lake. In this paper we desire to record our observations and conclusions, which seem to us important.

II.—LITERATURE.

Outside the Minnesota Geological Survey, the list of the investigators who have studied the geology of the Vermilion Lake area is not long.

Whittlesey, in 1876,* mentioned the occurrence, in the area about Vermilion Lake, of rocks resembling the Canadian Laurentian.

Chester,† in 1884, described the iron-ore at Vermilion Lake as occurring in connection with jasper and quartzite, intimately bedded with the country-rock, chiefly sericite schist, and standing in a nearly vertical attitude. The Vermilion iron-bearing rocks, and those of the Mesabi range, he says, are of the same age.

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* *Proc. Am. Assoc. Adv. Sci.*, Twenty-fourth Meeting, 1875, Part II., p. 60.

† *Eleventh Ann. Rept. Geol. and Nat. Hist. Survey of Minnesota, for* 1882, p. 160 *et seq.*

Willis gave, in 1886,* an interesting account of part of the Vermilion Lake rocks, illustrated by maps of his detailed observations. The area covered by these is nearly all included in that studied by us. He divides the rocks into seven formations, in ascending order as follows:

I. Light green, thinly laminated chlorite schist, exposed only at points on the anticlinal axes.

II. Jasper, made up of narrow bands of white, gray, brown and bright-red quartzite interstratified with layers of very hard blue specular iron-ore, which occurs also in ore-bodies of considerable superficial extent, and in fissures that run across the bedding. Thickness, 200 to 600 feet.

III. Chlorite schist similar to I. from which it is distinguishable only through stratigraphical position. Being softer than the quartzites II. and IV., between which it lies, its present thickness varies according to the pressure it has been subjected to. The original thickness of deposition was probably about 150 feet.

IV. Quartzite, banded in dark gray, white and black. It contains grains of magnetite which disturb the compass needle, but no ore. The thickness is probably about 200 feet. Beds I. to IV. form the ridges; V. to VII. are the valley-rocks.

V. Conglomerate of sandstone pebbles and bits of black slate enclosed in siliceous chlorite schist. The major axes of the pebbles lie parallel to the bedding of the schist. Between this and the next formation no very distinct line of contact was observed.

VI. Compact homogeneous rock composed of round quartz grains, chlorite, hornblende, plagioclase, and a little calcite. This is doubtfully considered a metamorphosed sedimentary bed of transition from V. to VII.

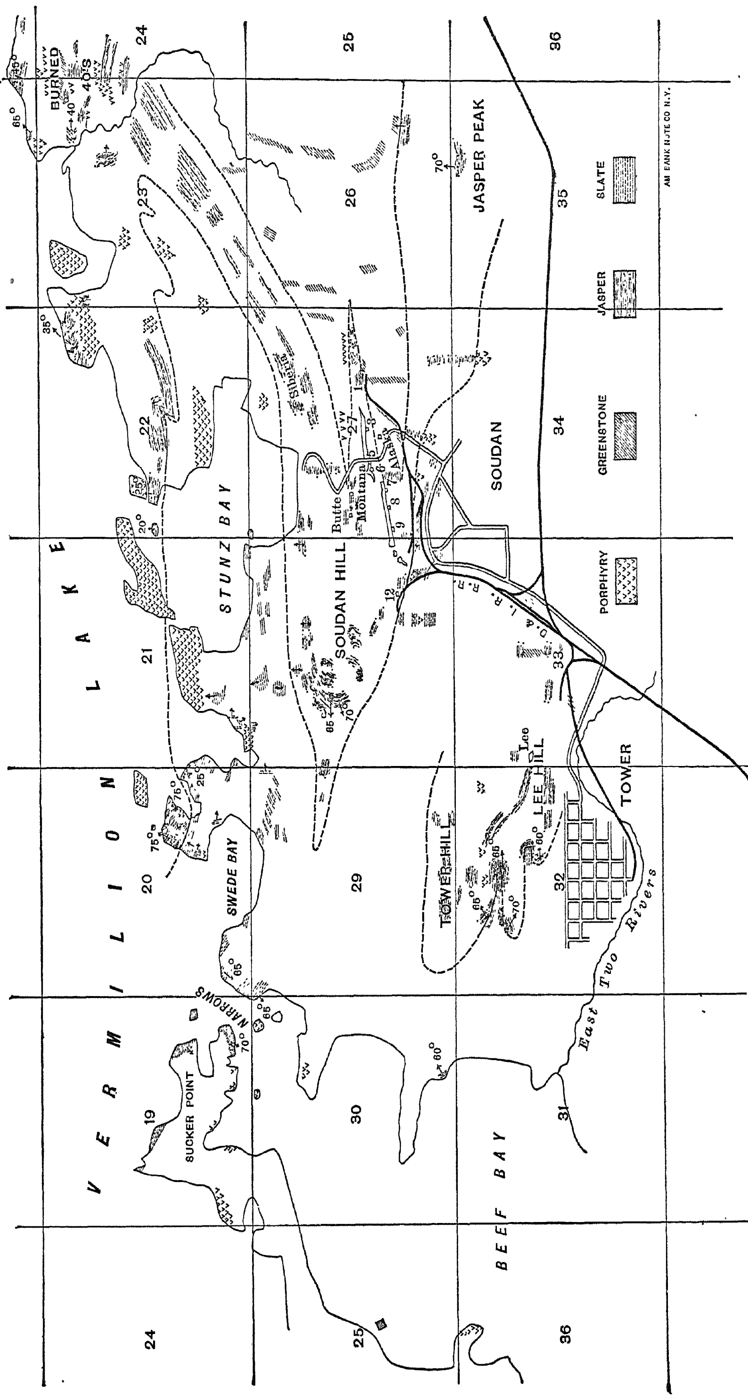
VII. Black clay-slate.

These rocks are folded into two anticlinal ridges, occupying what are now known as the Soudan and the Lee hills respectively, with a syncline forming the valley between.

This author recognized the existence of a sedimentary series, and that the structure is complex. His work is interesting for the further reason that it is the only serious attempt that has

* *Tenth Census Report,* vol. xv., pp. 457 to 467.

Fig. 1.



Geological Map of the Vicinity of the Tower Mines, Minnesota. The numbers are those of the sections, each one mile square.

been made to classify and to determine the order of succession in part of the Vermilion rocks. It was carried out in the only way in which these facts can finally be established, namely, by going over the whole ground, mapping all outcrops, and determining their strike and dip.

For the actual results themselves, but little can be said. The cleavage of the schists was mistakenly assumed to indicate their sedimentary origin. The "sandstone pebbles" are in reality quartz-porphyry. Formations I., III., V. and VI. are various igneous intrusives, which occur indiscriminately at any horizon. Formations II. and IV. are one and the same rock. Finally, the general succession is inverted, formation VII. being in fact older than the jaspers II. and IV.

Alexander Winchell, as the result of field-work done under the auspices of the Minnesota Survey in 1886, reached* interesting conclusions concerning the distribution, succession, and structure of the rocks of the Vermilion range.

> "In the western portion of the region the central part of the schist belt is occupied by strata predominately sericitic. With these are associated hematitic, magnetitic, and ferruginous jaspilitic beds and bands. . . . . The more chloritic schists lie on the north and south of the central line of the sericitic schists, forming two broken belts. . . . .
>
> "Generally speaking, the argillites are somewhat clearly restricted to belts still more removed from the main axis of the sericitic schists. . . . . Still outside of the predominant argillites, both on the north and south, are those forms of clastic rocks which I have styled graywacke."

This duplication north and south of the axial line was held to signify that these rocks occupy a single synclinal fold, and that the existence of more than one fold is an impossibility.

The conclusions of this author as to the relative geological positions of the argillites and jaspers are confirmed by our work.

R. D. Irving gave in 1888† a general account of the main belt of schistose rocks of northern Minnesota. In this he recognizes two types: the one, completely crystalline schists; the other, less completely crystalline. With the latter occur jaspers and ferruginous schists "whose identity as to nature and origin with the ferruginous schists of the iron-formation of the

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* *Fifteenth Ann. Rep. Minn. Survey*, pp. 174–175. See also *Bull. Geol. Soc. of America*, vol. i.

† *Seventh Ann. Rept. U. S. Geol. Survey*, 1885–86, p. 435 *et seq.*

south shore of Lake Superior and with the Animikie of the north shore, is complete." This series consists, in order of abundance, of schistose and slaty conglomerate, basic and acid eruptives, graywacke slates (at times carbonaceous), ferruginous and cherty schists, and quartzite; the whole many thousand feet thick. Folding and the development of a schistose structure by lateral pressure have here been pushed to the extreme.

Van Hise, in 1891,* divides the Vermilion iron-bearing rocks into a lower and an upper series, between which is an unconformity. At Vermilion Lake is a conglomerate containing abundant fragments of ore and jasper. This conglomerate is regarded by W. N. Merriam as a comparatively thin formation, overlying and overlapping the lower series. Merriam also found it to be thrown into a series of gentle folds, although having a vertical cleavage developed. The detailed evidence for these statements has never been published.

The Minnesota reports, which are the last to be considered, contain many details of the occurrence, bold speculations as to the origin, and but few conclusions as to the sequence and structure, of the so-called Keewatin rocks. In our area occur jaspilyte, "graywacke, argillyte, chlorite schist, greenstone, a syenite-looking massive rock, sericitic schist, agglomerates, porodyte, porphyritic massive rocks of various kinds, and fragmental conglomerates."† The bulk of these are regarded as volcanic sediments, the schistose structure being taken as the proof of sedimentation. Where massive crystalline rocks grade along the strike into schistose rocks of the same mineralogical composition, the schistose phase is held to be the original, and the massive to be derived from it by subsequent heat and pressure.‡ Other massive rocks are asserted to have been extruded in Keewatin time, and never since to have been "disturbed by oceanic distribution."§

With regard to the so-called conglomerates of Stunz Bay, it is clear that they are held to be such with doubt.

"About Stunz Bay, the Keewatin sediments have been made to act the rôle of eruptive rock. The rock having this igneous matter seems sometimes to be involved in the agglomerate which there prevails, somewhat like a dike, but

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* *Bull.* No. 86, *U. S. Geol. Survey*, p. 155.

† *The Iron Ores of Minnesota*, 1891, p. 25.

‡ *Ibid.*, pp. 41, 42, 43. § *Ibid.*, p. 26.

really wedges out in a lenticular manner. . . . . There is simply an abrupt transition from a schistose coarse rock with boulders to one without boulders, of the same color, massive or coarsely jointed, and homogeneous in mineral characters."*

These difficulties are compromised by regarding all the rocks referred to as submarine volcanic products.

"One of the most frequent, and yet one of the most perplexing facts in connection with these conglomerates, whether of acid or of basic pebbles, is the identity of the lithology of the pebbles with the strata in which they are embraced."†

The facts with reference to Stunz Bay conglomerates are truly observed, but are capable of a different interpretation.

The rocks most directly associated with the "jaspilyte" ‡ are stated to be sericitic and chloritic schist; which last has been seen to pass, on the one hand into argillyte, and on the other into lamprophyre.

A large number of details as to the relations between the sericitic schists and the "jaspilyte" are described and illustrated, for which the reader is referred to the Iron-ore volume of the Minnesota Survey.

The gist of the matter seems to be that, while the schists, for the most part, are interlaminated with the jaspers (*i.e.*, have their cleavage foliation parallel with, and as rock-masses have their longest dimension in the same direction as the jasper-bandings), yet frequently they cut across the latter, and hold numerous jasper inclusions near the contacts. They send tongues and branches into the jasper, and likewise include similar tongues of jasper and ore. In other cases, "the planes separating the great jasper-lodes from the country-rock are dis-

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* *Ibid.*, pp. 40, 41.

† *Ibid.*, p. 46.

‡ The word "jaspilyte" is not used in the sense in which it was proposed by Wadsworth, in 1880—as an acid eruptive rock, containing more than 80 per cent. of silica—but is applied to those rocks (regarded by the Minnesota Survey as direct chemical precipitates) which, made up of quartz mingled with the iron oxides in variable proportions, ordinarily go by the name of jasper, or iron-formation, in other parts of the Lake Superior region. In the present state of uncertainty concerning the real history of these rocks, the non-committal, though sometimes mineralogically inaccurate, word jasper is exceedingly serviceable; in the same way that the equally loose "greenstone" is useful in another field. It has the further advantage that it is in universal use among mining men. For the iron-formations of Lake Superior, the name jaspilyte is decidedly objectionable, not only because of its pigeon-English sound, but because its use might possibly lead to confusion in the future, when real jaspilytes shall have been discovered.

tinct; the transition is abrupt and complete in the narrowest possible space, the separated surfaces being very often slickensided, showing some movements in one mass in which the other mass did not participate." These varied and complex relations are explained by assuming that the materials of the jasper and of the schist were frequently deposited contemporaneously, and that the layers of each rock, as fast as they were deposited, frequently suffered immediate disruption from the unstable condition of the sea-floor, and from the violence of currents. The idea that the "jaspilyte" is eruptive is considered and rejected;* but the thought that the schist might possibly be eruptive does not seem to have been entertained at all.

The jasper is picturesquely described as

> "Made up of all possible varieties between pure silica and pure hematite. . . . . The silica and hematite bands run parallel to themselves, but are bent, retrorse, incurved, again revolute, abruptly broken and recemented, faulted minutely, any separate band independently faulted, or the entire mass suddenly jogged to the north or south."†

The jasper masses are regarded as lenses occurring in the schist without stratigraphic order, but grouped in greater frequency along certain belts. The banding of the jasper is exactly that of sedimentary beds.‡

The jasper and the ore-lenses, it is said, were formed at the same time that the enclosing rocks were deposited, and have undergone but very slight mineralogical or chemical modifications since that time; they were sometimes contemporaneously, sometimes successively, precipitated from solution in oceanic waters as silicic hydrate and ferric hydrate. Subsequent pressure alone was sufficient to dehydrate these oxides.§

No general structure is described, but the south range (Lee hill) is supposed to have been separated from the north range (Soudan hill) by a fault|| involving a heave of about one mile. All the disturbances of the area are referred to Archean time.

The general view of the Keewatin rocks entertained by the Minnesota geologists would, therefore, seem to be essentially the same as that held by Foster and Whitney long ago for the

* *Fifteenth Ann. Rep. Minn. Survey*, pp. 223–247. *The Iron-Ores of Minnesota*, pp. 230–237.

† *Ibid.*, p. 47.

‡ *Ibid.*, pp. 61, 62, 230.

§ *Ibid.*, p. 243.

|| *Ibid.*, p. 63.

iron-bearing rocks of Michigan, although the conclusion is not perhaps explicitly announced. That is, the Keewatin series is made up of sedimentary and igneous rocks, contemporaneously formed, subsequently folded, and now involved in inextricable confusion.

III.—The Sedimentary Rocks.

The rocks within the area which we have studied are partly sedimentary and partly igneous. The former belong to a series to which neither upper nor lower limits can yet be assigned, and to which it is, therefore, not unlikely that additional members, both underlying and overlying, may be added hereafter, outside of those here described.

The sedimentary rocks naturally fall into two divisions, which are sharply distinguished in appearance, composition and probable origin. The older of these is a slate formation, of unmistakable fragmental derivation, while the younger is an iron-bearing formation, lithologically identical with certain phases of the lower iron-formation in the Marquette district, and, to all appearance, quite devoid of clastic material.

The lower sedimentary member consists in the main of fine-grained argillaceous and siliceous slates, which often contain a small amount of carbonaceous matter and occasionally are very pyritiferous. In color they are black, gray, green or greenish-gray. Narrow bands of different colors and slightly different textures frequently alternate, giving the outcrop a finely-ribboned appearance. This structure, when present, does not generally coincide with the slaty cleavage, although it may do so; and since it arises from differences of material, there can be no doubt that it represents the real stratification of the rock. Very often, within the limits of outcrops, the slate is apparently quite homogeneous and shows no structure except the secondary cleavage. It is of interest to note that in thin sections representing four such outcrops, in every case a delicate banding at an angle with the cleavage is beautifully brought out as soon as the sections become thin enough to transmit light. The differences in the colors and textures of the little beds are so extremely slight as not to show in the outcrop or hand-specimen, even on a polished surface.

The coarser-textured varieties of rock included in the slate

formation are all, so far as we know, graywackes, and are composed of roundish to angular grains of quartz and feldspar set in a matrix of smaller fragments of the same minerals, together with a kaolinic substance, white mica, chlorite and calcite, the latter being alteration-products resulting from the decay of the feldspars. The graywackes usually occur as thin bands within the slates, and it is often difficult, and sometimes impossible, to distinguish them from the narrower intrusions of sheared porphyry. In one case, however, at the point north of Mrs. Ackley's boat-house, in Tower, in the S. W. ¼ of the S. E. ¼ of Section 30, a thickness of several feet of graywacke is exposed, which contains a layer of obscure conglomerate, holding pebbles that are apparently granitic. From its position, this outcrop, which is the coarsest fragmental rock found, may very possibly belong at a lower horizon than the other sedimentary rocks within our area.

The slates are always characterized by a more or less perfect slaty cleavage, which, while not entirely constant in direction, is yet sensibly so. In many, perhaps in most, localities it is altogether the most prominent structure, and so masks the bedding, which, however, it has in no sense obliterated. In certain cases, nevertheless, the bedding has been mechanically obscured in the following way: It happens that the cleavage-planes have often been surfaces of movement, along which the continuity of the rock has been broken, and when such surfaces traverse the bedding-planes, the latter are faulted, sometimes in the most minute and intricate way. In such cases we have the exact reverse of the *Ausweichungsclivage* of Heim. The rupture of the stratification-surfaces along little crenulations has not produced the cleavage, but slipping along the slaty cleavage has produced the appearance of ruptured contortions.

When we consider that these slates are among the most ancient known sediments, and that most of the conditions favoring metamorphism have been present in this region, their unaltered condition is a source of astonishment. They have been soaked with molten acid and basic magmas, and have been subjected to the action of mountain-building stresses probably at more than one period; and yet they remain, except for the slaty cleavage, almost without trace of metamorphism, and are essentially cleaved argillites, and not crystalline schists.

To what they owe this immunity is a matter for speculation. It has seemed to us that their poverty in coarser detrital material, and particularly in the feldspars, which are the principal source of the schist-making minerals, may have had something to do with it. Their chemical composition as aluminous silicate, with but little iron, lime, magnesia or alkalies, would be lacking in the elements necessary for the formation of new foliated silicates, while their extremely fine grain, and consequent imperviousness, would retard the introduction of these substances in solution from outside.

The slates outcrop along three main east-and-west zones. The most northern, as well as the most extensively exposed, of these stretches from the northern end of Sucker Point in Section 19 eastward to the northern shore of Stunz Bay, a distance of 3½ miles.

The second area lies on the southern slope of Soudan hill, the most easterly exposure being a little south of the Minnesota Iron Co.'s barn, in the village of Soudan, and the westernmost known occurrence being in a diamond-drill hole in the N. W. ¼ of the S. E. ¼ of Section 28. Between these are frequent outcrops, over a length of about ¾ of a mile.

The third area lies south of Tower hill and in the valley of East Two Rivers.

The rocks of the iron-bearing member are essentially composed of quartz, hematite and magnetite, variously mingled. Usually there is a definite segregation of these minerals into parallel bands, the thickest of which do not often exceed 3 or 4 inches. The quartz-bands are often white or uncolored; frequently they are black from intermingled magnetite or of various shades of brown or red from interspersed hematite. Besides the occurrence of the iron-oxides as coloring matter in the quartz-bands, these are found also in bands by themselves, in which quartz in various proportions is also present. No law of distribution of the two iron-ores, or of the variously-colored quartz, through the iron-formation, has been detected. Bands composed almost wholly of hematite are found in close proximity to magnetite bands, and white and red quartz-bands are often found side by side. It can be said, however, that the red quartz-bands are more usual and more brilliantly colored when the adjoining ferruginous bands are hematite. Probably the

most abundant variety of the rock is that in which the quartz is dark gray in color and the interleaved ore-bands are thin and largely mingled with quartz.

The banded structure is, for the most part, exceedingly regular and persistent, and, as N. H. Winchell has said, is clearly the structure of a sedimentary rock. The individual layers are usually continuous within the limits of exposures. In one case a single band of brilliant red jasper was followed for 135 feet. By the eye and under the microscope the rock cannot be distinguished from similar varieties of the Lower Marquette iron-formation. The quartz varies greatly in fineness, from minutely to coarsely crystalline. It usually occurs in distinct linearly-bounded grains, which are never rounded. These grains are often dusted through from center to perimeter with little opaque or red dots, which, under the highest powers, show the crystalline form of magnetite or hematite. There can be no doubt that this quartz has crystallized in place. The magnetite when detached shows sharp crystal outlines, while the hematite occurs in irregular aggregates or in single flattened rhombohedra.

None of these minerals, therefore, can be mechanical sediments in their present form; and yet the structure of the rock which they together make up is eminently a sedimentary structure. Unless, then, we adopt the purely speculative view of the Messrs. Winchell that the materials here present were originally precipitated from solution in the waters of the ocean, we must conclude that the rock is one that has undergone profound chemical rearrangements with probable modification of its original composition. From the slight microscopical study that we have been able to give to the Vermilion jaspers we cannot say what the character of the original rock was. Nothing farther back than the jaspers has been found. It can be said, however, that the present minerals represent ultimate products of oxidation under certain conditions, and so in themselves doubtless indicate the nature of the change that has taken place. From the analogies of the other iron-districts of the Lake Superior region, where entirely similar rocks are found as the end-products of a series of changes due to oxidation, which lead back, on the Gogebic range,* to a cherty iron

* R. D. Irving and C. R. Van Hise: *Monograph xix., U. S. Geological Survey.*

carbonate, and on the Mesabi range,* at least in part, to a glauconitic green-sand, we believe that here, too, we have had like changes, starting from like original rocks of one or both of these types; but that here the intermediate stages have been entirely passed and the whole rock has reached the end of the process.

The jaspers show the most extreme deformation as the result of mountain-building stresses. While they undoubtedly have not suffered more than the slates, yet their more conspicuous bedding makes these effects much more striking in them. It is the rule to find the jaspers contorted and closely folded. Frequently, over considerable areas, the point of rupture has been passed, and the rock is now a broken mass, in which the units stand in every conceivable attitude (Fig. 2). In the Soudan syncline this is especially true; and consequently strikes and dips have here but little determinative value. The little folds into which the jasper has been thrown have no constant direction of pitch. They often stand vertical or incline in opposite directions at points closely adjacent. Pumpelly's law, viz., that the degree and direction of pitch of a large fold are often indicated by those of the axes of its subordinate plications, therefore, does not hold good here. The reason for this appears to be that the corrugating forces have acted, not in one direction only, but in two. At some indeterminate time, subsequent to the formation of the principal folds, this region was subjected to stresses nearly at right angles to the earlier, or parallel to the axes of the folds already formed. Excellent examples of the effects produced in the jasper by this end-for-end compression may be studied in the neighborhood of Siberia scram and also near the western end of the Soudan syncline, where, indeed, not only the subordinate folds pitch east and west, north and south, but the whole syncline has suffered a local overturn.

The jaspers occur also in three principal east and west belts. The northernmost is exposed at intervals over a length of nearly three miles, from the east shore of Swede Bay to the "burned 40's" in Section 23. On the north it dips under the waters of Vermilion Lake, while to the south it is bounded by the principal belt of slate.

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* J. E. Spurr: *Bull. x., Geol. and Nat. Hist. Survey of Minnesota.*

The next area of jasper to the south is much the largest of the three. It begins in Section 28 as a narrow tongue, which gradually widens in going east until it reaches a maximum, on a north-and-south line through Montana shaft, of nearly half a mile. East of Montana, this area is divided into three parts by tongues of igneous rock coming in from the east. The northernmost continues to the northeast as far as the "burned 40's," where it joins the jasper of the northern belt. The middle finger has an extent of only about a quarter of a mile to the east, finally tapering to a point in the irruptives; while the southern fork has a known length of over one mile beyond the line of divergence, and undoubtedly a far greater easterly extension.

The third belt of jasper is that which forms Tower and Lee hills, in Sections 29, 32 and 33, and occurs immediately north of the southern belt of slates.

IV.—The Igneous Rocks.

Most intimately involved with the two sedimentary formations is a great volume of igneous rocks, both acid and basic, which occur for the most part interleaved with the former as intrusive sheets, but often cut their original lamination as dikes. The individual sheets of igneous material range from the width of a knife-blade to masses hundreds of feet in thickness. They occur in numbers so great that the aggregate certainly exceeds, and perhaps reaches several times, the combined thickness of the sediments.

These igneous rocks present the most varied aspects in the field, in consequence chiefly of the different degrees to which they have yielded to mechanical forces, and of the different stages of alteration which their original minerals have reached. At the one extreme their igneous character is evident at a glance, while at the other it can be determined only by the field-relations, or inferred by tracing back the processes of alteration step by step. The recognition of the true character of the obscurer forms is the key to the structural geology of the Vermilion range.

The acid rocks, so far as our observation goes, are now, or were originally, for the most part quartz-porphyries. In the less altered occurrences, they appear in the field to be made up

of a white weathering matrix in which roundish grains of bluish and glassy quartz, of nearly uniform size, and crystals of white feldspar are pretty uniformly distributed. The weathered matrix possesses the opaque white color characteristic of rocks into the composition of which feldspar largely enters. The fresh fracture is much darker, and is gray in color, often with a tinge of green. Occasionally the porphyritic quartz and feldspar crystals are present only sparingly, and the rock is mainly made up of the ground-mass alone. On the other hand, but more rarely, phases occur in which the rock breaks with a granular fracture, which gives brilliant reflections from numerous cleaved surfaces of feldspar, and the texture appears to be wholly granitic. Other differences also arise from the relative abundance of the quartz-phenocrysts, which occasionally are rare, though seldom entirely absent, and from the occasional presence of hornblende with crystal boundaries.

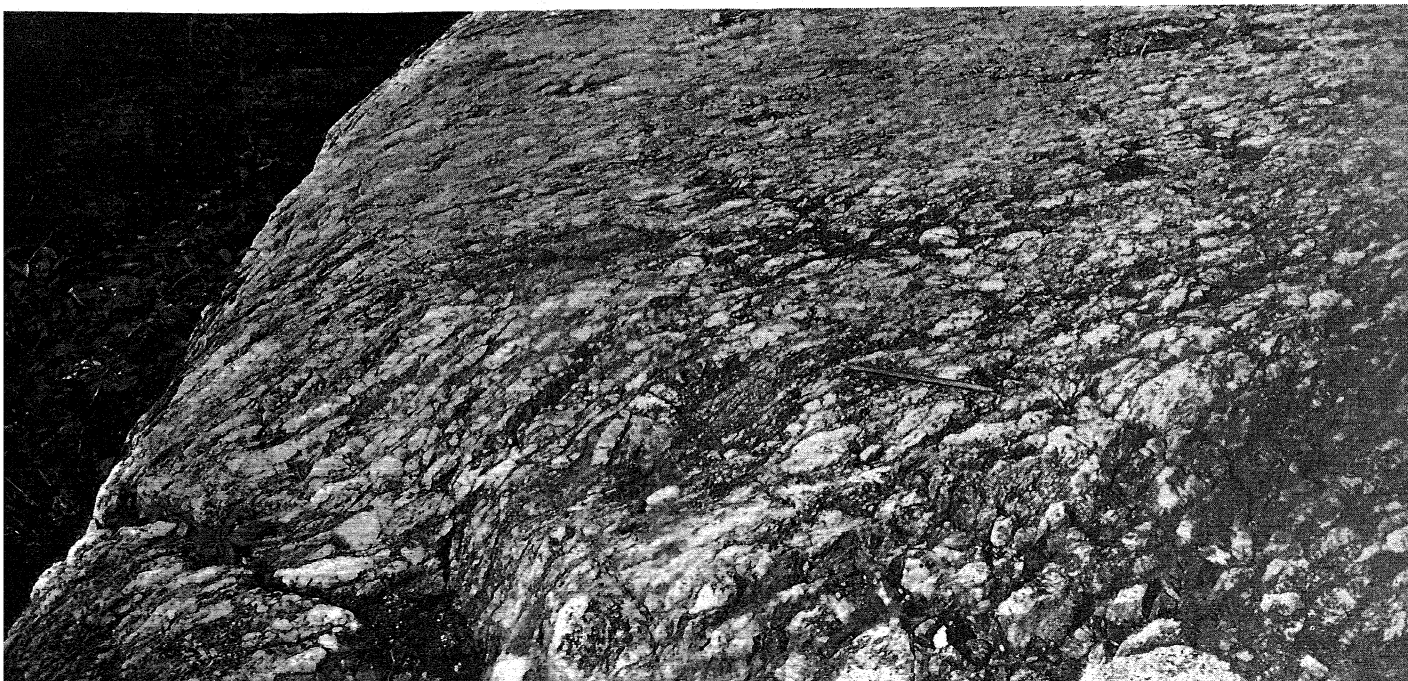
In thin section, the ground-mass of these rocks is in all cases holocrystalline; but the variations in coarseness and in the space which it occupies relatively to the phenocrysts are very great. The phenocrysts are quartz, orthoclase, plagioclase and more rarely hornblende; and all are found in idiomorphic forms. The quartzes occur in double pyramids, which often show corrosion by, and inclusions of, the ground-mass. The plagioclases, which were not specifically determined, never give large extinction-angles, and no doubt belong at the acid end of the series. Micropegmatitic intergrowths of quartz and feldspar were observed in some sections, in which cases the rocks become granophyres. Calcite, sericite, chlorite and quartz occur abundantly as alteration-products.

The more massive occurrences, however, are but relatively massive, for they never fail to show more or less plainly the effects of deforming forces; only, these effects are less conspicuous than in the varieties to be described. They are comparatively rare, and of small extent, and from their manner of occurrence are evidently cores which, from causes that cannot be precisely assigned, have escaped the extreme effects that were produced in the same rock-mass all about them. Two very distinct types appear as the ultimate results of the application of mechanical forces to the original quartz-porphyries. Between them and the least affected portions there are number-

FIG. 2.

Jasper Breccia, Exposed in the S. W. Quarter of the N. E. Quarter of Section 32.

Fig. 3.



Porphyry Conglomerate-Breccia, Exposed in the N. E. Quarter of the S. E. Quarter of Section 20.

Fig. 4.

Porphyry Conglomerate-Breccia, Exposed on the Island near East Quarter-post, Section 20.

Fig. 5.

Contorted Jasper, Exposed East of Siberia Scram (Looking West).

less gradations. These end-products are, on the one hand, sericite-schists, and, on the other, conglomerate breccias.

The sericite-schists are close-textured light-weathering rocks of characteristic silky luster, and a more or less perfect schistose structure, which has the usual east-and-west direction, like the cleavage in the slates. Even when most perfectly schistose, these rocks usually show little eyes of quartz, quite evenly distributed, which can often be seen with the lens to be entirely crushed or to be surrounded by granulated margins. These quartzes, in size and in their manner of distribution through the rock, are exactly like the quartzes of porphyries. And in their relations with the slates and jaspers, the schists frequently show the clearest eruptive contacts.

Under the microscope it can be proved, in most instances, that they are in reality sheared and crushed quartz-porphyries. We have, on the one hand, numerous cases in which the thin section shows a quartz-porphyry traversed by a few irregular and generally parallel lines of fracture, along which a little chlorite and sericite have grown, but chiefly brought out by the infiltration of foreign material, such as iron oxide or carbonaceous matter. The phenocrysts, while strained and often broken, retain their crystal boundaries, and the ground-mass between the fractures is unaltered, save for the presence of little grains of calcite and minute plates of sericite, which are often disposed in two parallel directions. The further development consists in the increase in number of the fractures, with a resulting increase in the area occupied by the new minerals, and in the progress of the crushing and fracture of the phenocrysts. The feldspars are the first to go, while the quartzes, even in the most schistose varieties studied, although granulated to a fine mosaic, are easily recognized by their characteristic distribution and size. In nearly all cases some portions of the ground-mass, with the parallel arrangement of the little plates of sericite, remain between the lines of fracture.

The schistose structure in these rocks is therefore not an original structure, depending upon any arrangement of the primary constituents, but dates from a time subsequent to the time when these were formed. It has been produced by the growth of secondary minerals which have developed at the expense of the original minerals, for the most part in spaces which have

been mechanically determined by the rupture of the original rock along roughly parallel lines. The structure is therefore in no way related to bedding; and the question, whether the rocks which possess it were ever bedded, must be decided on entirely different grounds. That they never were bedded by water-deposition, cannot be doubted for a moment by any one who studies these rocks in thin section, or who understands the significance of irruptive contacts in the field.

The conglomerate-breccias constitute the second type of rock mechanically derived from the original porphyry form. These, when most perfectly developed, consist of elongated inclusions of quartz-porphyry with rounded outlines, closely packed in a similar porphyritic matrix, which differs from the inclusions only in being slightly schistose. The pebbles are aligned with their long axes pointing in the same direction. On the weathered surface the harder inclusions stand out above the matrix, which usually has also a darker color from the frequent presence of chlorite, and infiltrated coloring-matter along the schistose partings. On the freshly fractured surface these differences of color largely vanish, and the pebble outlines are much more indefinite. The rock then appears as a lithological unit, through which intersecting zones of schistosity pass, representing the matrix, and inclosing areas without schistose structure, representing the pebbles. The pebbles and the matrix are alike porphyritic, and carry similar phenocrysts (Figs. 3 and 4).

In limited outcrops the likeness to conglomerates built up of water-worn pebbles of quartz-porphyry is frequently very striking. Suspicion is roused by the fact that the pebbles are practically of the same rock; by their close uniformity in size and in the elongated form; by the very sparing matrix; and by the fact that the matrix has as clearly marked an eruptive character as the pebbles and belongs to the same rock. No layers of finer material are ever present, and there are but slight variations in lithological character. The rock bears no marks of stratification, unless the general parallelism of the inclusions is taken to be such. In other cases, however, the resemblance to water-deposited rocks is greatly strengthened by the presence of fragments of jasper or slate. These occur in all sizes, from masses 80 or even 100 feet in length down to little pieces the

size of a pea; of course, the smaller fragments only can be mistaken for true pebbles. Slate and jasper fragments, so far as we have observed, never occur adjacent to each other, but each variety is confined to the neighborhood of contacts with similar rocks. The jasper inclusions frequently come in along the same line for several feet, almost or quite in contact with each other, and then suddenly cease, suggesting in their relations to each other that they originally composed a single band which has been broken up. They generally have sharply angular contours, often with re-entrant angles, although very perfectly rounded forms are not entirely lacking. Finally, in a few cases, jasper inclusions were observed to be themselves entirely or partly contained within quartz-porphyry inclusions. The slate inclusions are always angular, and penetrate the inclosing porphyry in delicate feathery edges. The best examples are to be seen in the great sheet of porphyry which faces the lake in the southern part of Section 21. This sheet carries only slate inclusions which have a wide range in size. Near the water's edge, about 100 paces west of the N. S. ¼ line of the section, an enormous fragment of slate, measuring 120 feet in length, can be seen almost entirely buried in porphyry, which, besides, sends little tongues into it. In the eastern half of Section 23 similar relations are even better displayed between jasper and porphyry. A careful study of these localities leaves no room for doubt that the slate and jasper inclusions are fragments plucked off by invading igneous sheets from the slate and jasper formations.

The next step back from the most perfectly developed conglomerate structure is that in which the quartz-porphyry inclusions, instead of having rounded contours, have rectilinear boundaries corresponding in adjacent inclusions, which are separated from each other by very narrow zones of schistosity. These zones, while irregular in detail, follow two main directions, which intersect at an acute angle, the bisectrix of which has a nearly constant direction, substantially parallel to the cleavage in the slates, or about east-and-west. In this phase the inclusions no longer have the pebble-form, but, on horizontal and vertical sections, appear as more or less perfect rhombs, with slightly rounded corners. The included blocks are often traversed by irregular cross-fissures, which in some cases still

stand open, but usually have been filled with vein-quartz. These, evidently, have been caused by direct pulling apart, and are good evidence of the complex strains that have existed within the rock.

Finally, in the initial stage we find massive quartz-porphyry traversed by two principal sets of surfaces of parting (on horizontal and vertical sections), which intersect at acute angles and divide the rock into sharply-angular blocks. The dividing planes exist not only as actual partings, upon which weathering seizes to degrade the rock, but also as tendencies to break, and then constitute a kind of cleavage. The intervals vary widely, the included cores ranging from several feet down to an inch or less in diameter.

The angles of intersection of the main systems of parting are not constant, while the direction of the bisectrix, as shown by the parallelism of the inclusions with the cleavage of the slates, is sensibly so. The acuter angles seem to accompany the division of the rock into the smaller blocks. A good example of the finer division, in which the dividing-surfaces are unusually regular and evenly spaced, may be seen on the low island in the N. E. ¼ of the N. E. ¼ of Section 30. The photograph (Fig. 4) fairly represents the average coarseness of the structure. It was taken at the west end of the large island in the eastern part of Section 20, and shows partial rounding of some of the larger forms. At the left, in this photograph, the intersecting partings are seen as fine, sharp lines. It will be noticed that some of these lines penetrate part way into a core and then die out; that is, all are not equally strongly developed.

We thus have a continuous series, often to be seen within the same outcrop, from quartz-porphyries entirely massive (save for the presence of intersecting parallel partings), through occurrences in which the parallel partings are represented by zones of schistosity, to those in which these zones become more prominent and include pebble-like forms of the unaffected rock. In the final stage the resemblance to detrital conglomerates is often extremely close, and for such these breccias have generally been taken in the Stunz Bay area, where they are largely exposed and convenient of access. That they are, however, not detrital rocks, but, on the contrary, merely mechanically-derived phases of massive igneous rocks, is abundantly clear. In a later sec-

tion we shall consider in some detail the origin of this interesting structure.

The porphyries are everywhere abundant in one form or another. The sericite-schists, perhaps, are the prevailing type where the sheets are thin, particularly within the synclinal jasper-areas. The more massive varieties with the conglomerate breccias are, in the main, characteristic of the greater laccolites, and have their typical occurrence within the anticlines. But to both rules—if, indeed, they can be called such—there are numerous exceptions.

It is quite impossible to represent upon the scale of the map all the thin sheets, and equally impossible to trace them out upon the ground. It need only be remembered that within the jasper and slate areas they occur by the hundred, for the most part evenly interbedded with the sediments. Only the larger ones can find a place upon the map.

Of these the great laccolite in Sections 20, 21 and 22 possesses especial interest, as it furnishes clear proof of the intrusive character of the rock and of the mechanical derivation of the sericite-schists and conglomerate breccias. In Section 21 this sheet has an exposed width of about 400 feet from north to south, and occupies the headland in the southern part of the section continuously for $\frac{5}{8}$ of a mile. The large island across the mouth of Stunz Bay, which lies partly in Section 21 and partly in Section 22, clearly belongs to the same mass, and is separated from it by a narrow stretch of water. Still farther east it is probable that the smaller islands and the headlands of porphyry in the central and eastern portions of Section 22 also have an under-water connection with it. To the west the porphyry-exposures on the two headlands in Section 20, and on the two islands north of them, are most likely portions of the same mass. If all are included, its extreme length from east to west is nearly $2\frac{1}{2}$ miles, while its breadth from north to south cannot fall short of a quarter of a mile.

That portion which forms the headland in Section 21 is limited on the south by the main area of slates, which strike about east and west and dip towards the north at high angles. Contacts with the slates are not exposed; but the general trend of the southern boundary of the porphyry mass agrees closely with the observed strikes in the neighboring slate outcrops. The

porphyry carries, however, a large number of slate fragments, from pieces of the size of a pea upwards, all of which are angular. The smaller fragments are noticeably hard and brittle. Under the microscope they are seen to owe these characters to silicification. They have been soaked through with new finely-crystalline quartz which has penetrated the mass in a fine network of delicate threads.

The largest slate inclusion has already been referred to.* This fragment is about 12 feet wide at its widest, and is exposed for a length of 120 feet. At the eastern end it plunges into the porphyry with a pitch of about 20°, and is surrounded at the two sides and above by the igneous rock; to the west it gradually tapers to a point in the porphyry, and thus is seen to be limited in all directions save one. The bedding of the block is in sharply contrasted light and dark colors, and is very distinct. The strike of the color bands is about north and south, or with the shorter horizontal dimension, and the dip is with the general pitch, about 20° to the east. There is, besides, a vertical and very perfect slaty cleavage, which strikes, as usual, east and west. At the eastern end, the contact with the porphyry can be studied closely, and the eruptive is seen to penetrate the slate in numerous small tongues, both parallel and transverse to the bedding. The occurrence of this large fragment so far from the base of the sheet probably indicates that it has been derived from above, and that therefore under water the sheet has a slate boundary not far to the north.

The large island across the outlet to Stunz Bay is separated from the area just described by a narrow strait less than 100 feet in width. It also is made up of porphyry, mostly of the conglomerate-breccia type, which, in the eastern portion especially, holds many fragments of jasper, and none of slate. At the southwest end of the island the porphyry is in contact with slate just at the water. South of its eastern end the little island near the west quarter-post of Section 22 is entirely composed of jasper, as is also the nearest point on the mainland to the east. The southern edge of this mass of porphyry has therefore, within the length of the island, crossed from the slate formation to the jasper, the line of contact between the two last

* Pages 610 and 611.

passing through the southern part of the island. The reason for the change in the character of the inclusions from slate to jasper in going from west to east, is therefore readily understood.

The porphyry sheets in Section 20 are not so certainly part of the same mass, as there is a water-covered stretch of a quarter of a mile between it and the nearest of them. But they belong at least to a closely related sheet.

On the east line of Section 20, the outer side of the point is entirely made up of granite porphyry, which, at the west end, is in visible contact with slates. These strike locally between north and northeast and have a low southeasterly dip. The porphyry cuts the strike of the slates nearly at right angles, sends apophyses into them, and carries slate fragments. Its intrusive character is therefore clear.

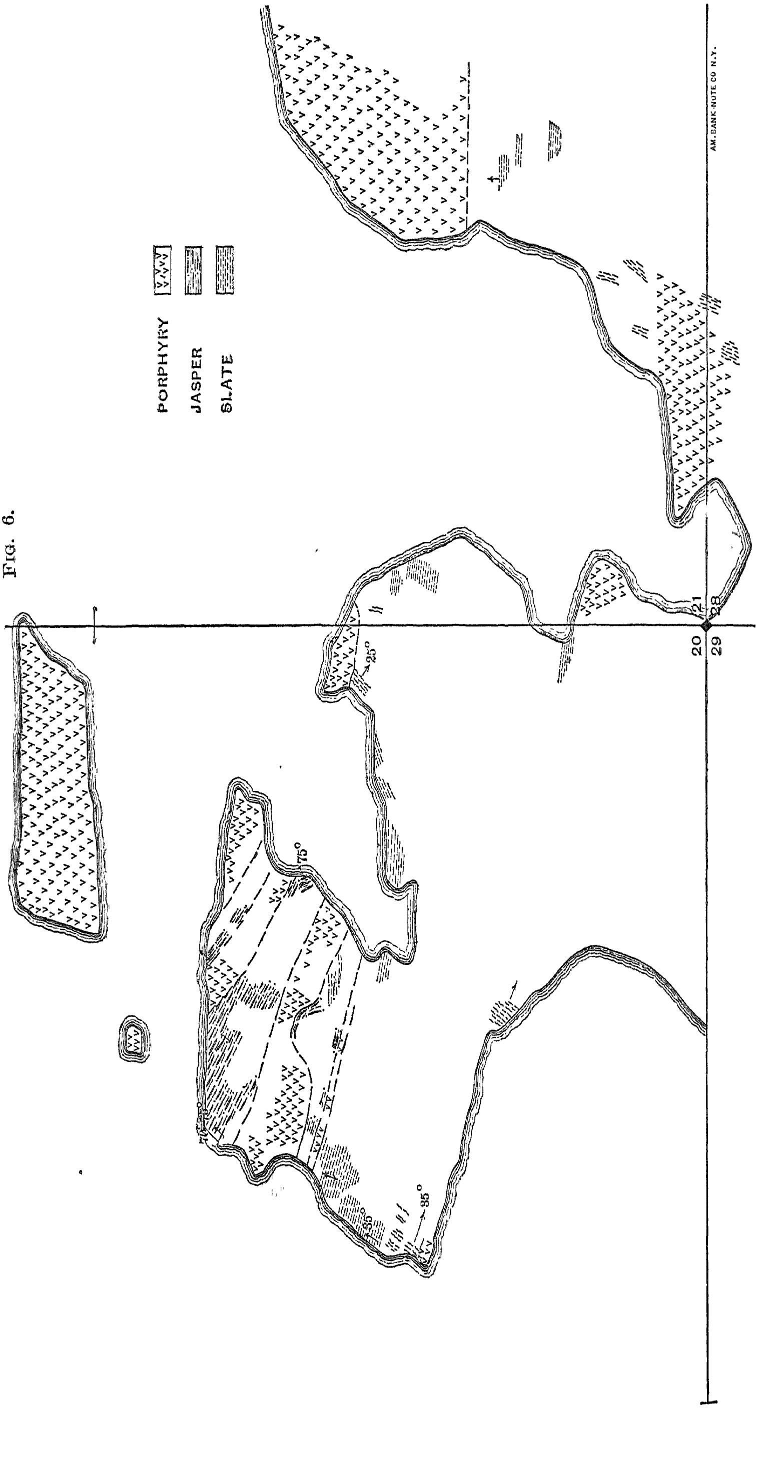
The island near the east quarter-post of Section 20 is made up wholly of porphyry, with many jasper inclusions; and here the beginning of the conglomerate-structure is well shown.

The point jutting into the lake in the eastern half of Section 20 is underlain by jasper and porphyry in the northern portion, and by slates and porphyry in the southern. The relations are shown on a larger scale on the map, Fig. 6.

It will be seen that the jasper is here split into three parallel belts by intervening belts of porphyry. The jasper has in the main a uniform strike to the north of west, with a dip to the east of north, of 70° and upwards. Between the northernmost belt of jasper and the porphyry which forms the eastern extremity of the point no contacts are visible. The next porphyry belt may be seen, on and near the outer shore, in contact with jasper on both sides. The contact-surfaces stand at a high angle, and, for the short vertical interval through which they are exposed, are strictly parallel with the dip of the jasper. The southern boundary of this belt traverses the strike of the adjacent jasper in steps, eating south into the mass in going east. Within the porphyry many fragments of jasper are included; but, although the rock is greatly fractured in the usual directions, the conglomerate-breccia structure is not very noticeable. There can be no doubt that this porphyry belt has been intruded into the jasper, nearly but not quite parallel to its bedding.

The main porphyry belt succeds the second zone of jasper,

FIG. 6.



Geological Map of the Outcrops on Sections 20 and 21 (see Fig. 1), Showing the Relations of the Intruded Porphyry-sheets to the Jasper and Slates. Scale, 4 inches = 1 mile.

with which no exact contacts are uncovered, although the rocks in several places are only a few feet apart. In this belt we find all the phases of massive, brecciated and conglomeratic porphyry, and sericite schists. The belt is widest at the west end, where the exposures near the shore are large; and for the most part, the rock has not been mechanically affected to any great degree, although the usual surfaces of parting in two directions are distinct. Along the southern margin it terminates against a narrow and disturbed belt of jasper. Near this jasper the porphyry becomes schistose, and includes angular fragments and layers of jasper.

In tracing the zone of porphyry to the east it becomes much narrower and altogether schistose, while the jasper belt to the south correspondingly widens. Near the middle of the point both sericite-schist and jasper are well exposed. The jasper, back from the contact, is greatly contorted, and the strike in the layers exposed turns from north of west to almost due north and south. In approaching the contact the jasper layers become broken and faulted in the most intricate way, while the contact-zone itself is one of extreme brecciation, in which fragments of both rocks are involved. Then follows the very perfect sericite-schist.

We have here direct field-evidence of the derivation of the sericite-schist from porphyry by shearing. The fragments of porphyry in the breccia, as well as those of jasper, are frictionally rounded, and in some cases rather coarsely fractured and pulled apart; but not at all schistose. The breccia consists wholly of such porphyry and jasper fragments cemented by the flour of attrition. On the one side stands the solid jasper which has supplied the jasper fragments; while on the other is the sericite schist. The inference that the schist before the movement which produced the breccia was also solid porphyry can hardly be avoided, and is entirely confirmed by the study of the microscopic sections.

The eastern prolongation of these two belts towards the little bay shows a gradual encroachment of the schists on the jasper by a deflection of the contact-line to the south. At two localities visible contacts are seen, in which the schists cut squarely across the bedding and hold numerous fragments of the jasper. Before the bay is reached the schists gradually pass into very

perfect examples of porphyry conglomerate-breccias, which also outcrop just at the shore.

This porphyry zone is a very instructive study. It is a continuous belt separating two portions of the jasper-formation, which stand almost on end; and with these its relations are those of an intruded sheet. Within the belt, and, indeed, within limited outcrops in it, we have both the original unaffected porphyry and all its various mechanical derivatives.

Contacts of the southern jasper belt, which is the narrowest of the three, with the underlying slates are not uncovered; but there is perfect correspondence in strike and dip between them. In two or three places a narrow belt of schistose porphyry, which carries both slate and jasper inclusions, intervenes and is probably continuous.

These four intrusive sheets of porphyry do not continue into the slates on the eastern side of the little bay. There can be little doubt that they join under water with the sheet exposed on the opposite point, and so are westerly diverging fingers from the southern margin of the great laccolite in Sections 21 and 22. It is not impossible that the porphyries represented by the very perfect conglomerate-breccias of the eastern shore of Stunz Bay are also branches from the same mass.

From Stunz Bay eastward to the "burned 40's," porphyry outcrops are very abundant, and numerous contacts with the jasper of the northern belt are uncovered. These display penetration of the jasper by the porphyry, inclusions of the former in the latter, and in general the phenomena of igneous contacts. The field-evidence through the northern portions of Sections 22 and 23 leaves nothing to be desired to prove that the porphyry has been injected into the jasper in the form of intrusive sheets.

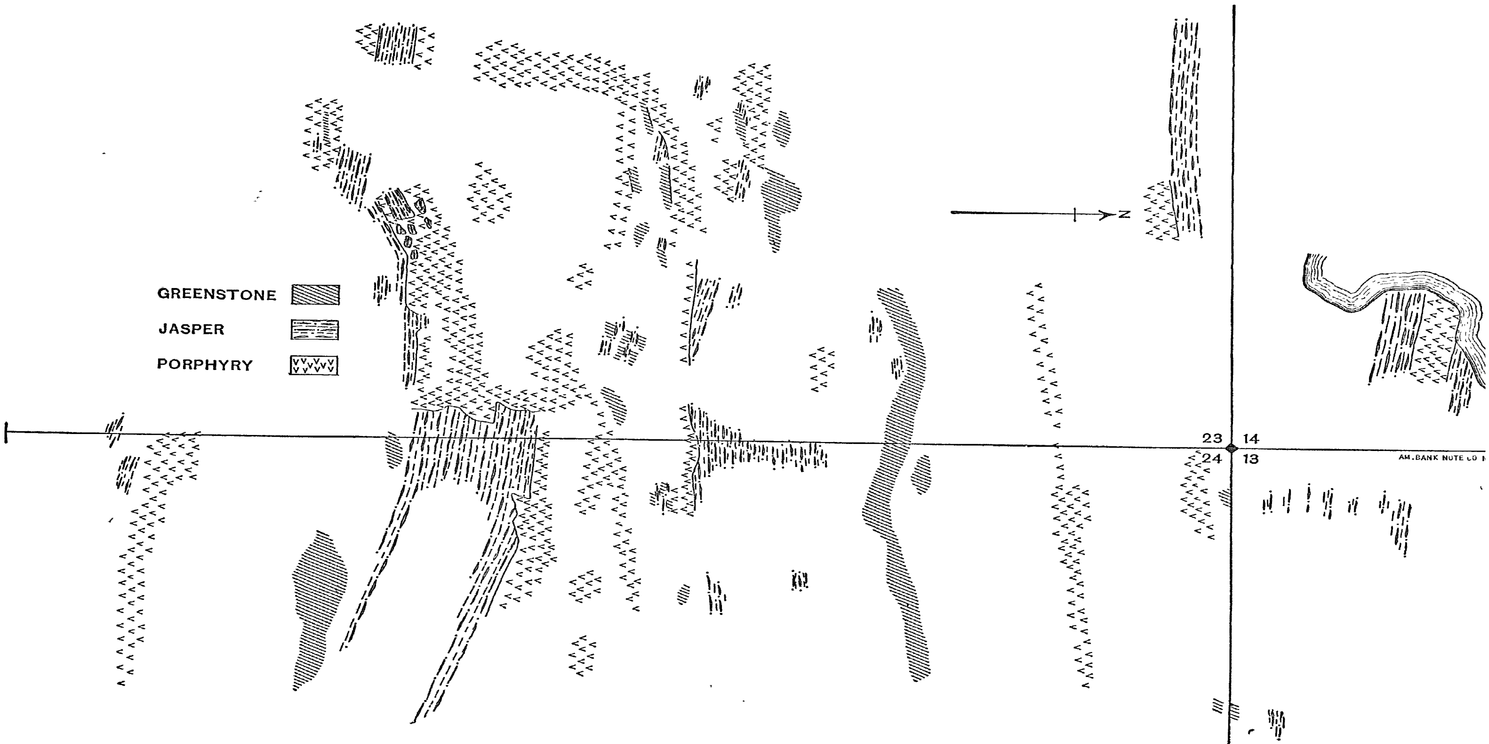
Description cannot be multiplied indefinitely; but it is worth while to mention the beautiful exposures in the "burned 40's," which reveal the relations of the two rocks with perfect clearness. Here, for a distance across the strike of nearly a half mile, the larger part of the rock-surface is exposed. The only bedded rocks are the jaspers, which, in the southern portion, strike a little south of east and stand nearly vertical; while, in the northern portion, they strike somewhat farther to the southeast and dip to the northeast. In and between the

bedded rocks are intrusives. The earliest are schistose greenstones, which penetrate and hold fragments of the jasper. These, again, with the jasper, are cut by the greatest profusion of porphyries and their mechanical derivatives. The relations are fully illustrated on the map (Fig. 7), which renders verbal description unnecessary. It need only be said that the porphyry has penetrated between the bedding-planes of the jasper; that frequently it also cuts across these planes; and that it includes jasper fragments up to the size of a house. The most beautiful examples of sericite-schists, conglomerate-breccias and relatively unaffected porphyry may here be seen over and over again within the limits of the same rock-exposure.

The greenstones display all the phases already described in the case of the porphyries, viz., massive, schistose and conglomerate-breccias. In the field the more massive varieties are dark brownish to green-weathering rocks, without evident quartz, soft superficially to the pick-point, but excessively tough. On the fresh fracture they usually show a medium grain, with a general greenish-gray color, in which appear lighter areas of feldspar occasionally fresh enough to show multiple twinning and darker greenish-black spots. Of rather exceptional occurrence is a hard greenish-gray aphanitic variety, the freshly-fractured surface of which displays now and then a glint of feldspar. Under the microscope these rocks are greatly altered; but the feldspars are usually recognizable and are predominantly plagioclase. Except the usual accessories, magnetite, apatite, etc., the other minerals are all secondary, and comprise chlorite, actinolite, green hornblende, epidote, calcite, zoisite, quartz and rutile. The characteristic diabase-structure is often very clear.

From the massive varieties the same mechanical derivatives appear as in the case of the quartz-porphyries, the end-products being chlorite- and actinolite-schists, and greenstone conglomerate-breccias. The field evidence for the derivation of these from originally massive greenstones is quite as clear, and is of the same character, as for the analogous porphyry-forms. But for the chlorite-schists it has not, in the same degree, corroboration from microscopic study. In the more perfectly schistose forms all the original minerals have disappeared; and the slides in themselves give no clue to their history. The field-evidence,

Fig. 7.



Geological Map of the Outcrops in the "Burned Forties." Scale, 13 inches = 1 mile.

however, is so convincing that absolute proof from the microscope may be dispensed with.

Like the sericite-schists, the chlorite-schists are mainly found in the narrower sheets and in the synclines, the massive and conglomeratic forms being usually characteristic of the anticlinal areas. To this rule there is an important exception. The great greenstone sheet which reaches the surface on the line of the Soudan syncline is in the main massive, with numerous partings and locally developed conglomeratic structure, while in the fingers in which it penetrates the jasper on the west it passes into very perfect chlorite-schist. This sheet, which is known to occupy the greater part of Sections 25 and 26, and has a great (as yet undetermined) eastward extension, is by far the largest single mass of greenstone found. Other important areas occur in the "burned 40's," where they are older than the porphyries, in the large exposures in the Soudan railroad yard, and on the little rise of ground east of Lee hill. Good examples of greenstone conglomerate-breccia occur just at the center of Section 32.

V.—The Geological Structure.

The most northern and the largest of the three areas of slate extends from Sucker Point, in Section 19–62–15, for 3½ miles along the southern shore of the lake, to the north shore of Stunz Bay in Section 22–62–15, and occupies a belt about half a mile in width. The lake-shore, at the narrows in Section 19 and at Stunz Bay, cuts completely across this belt, while Swede Bay, and the smaller bays in Sections 20 and 21, have gnawed more or less deeply into it. Between Swede Bay on the west and Stunz Bay on the east, the slates are well exposed inland on many low east-and-west ridges.

From the most westerly exposure on the eastern shore of the narrows near the corner of the four Sections 19–20–30–29, the slates, as far as the expansion into the main lake, have a constant northwesterly strike, and a dip to the south of about 65°, while the slaty cleavage strikes nearly east and west, and stands nearly vertical. These conditions of strike and dip continue as far as the south and east sides of Swede Bay. At the outcrop near the north line of Section 29 on the south shore, the rock is greatly contorted, and the real attitude is not de-

terminable. On the north shore of Swede Bay, the strike is about north and south, and the rocks lie nearly flat, but show on the whole an easterly pitch, although low inclinations to the west are also to be seen. In going north along the western shore of the point in the S. E. $\frac{1}{4}$ of Section 20, the strike quickly changes to the northwest, and for a short distance the slates dip north at a low angle, which rapidly increases, until, at the northernmost exposures, the dip is nearly vertical. Finally, the slates are succeeded by jaspers of the northern jasper belt, also striking northwest and dipping northeast from 70° to 75°; and these, with the intercalated porphyry sheets, occupy the northern half of the point. In the excellent section across the strike afforded by this shore-line, the northern slate belt has therefore an unmistakable anticlinal structure. The axis of this anticline may be traced east into Section 21, and is indicated also in the N. E. $\frac{1}{4}$ of Section 29 by the gradual bending to the north of the contorted strike.

The numerous outcrops in the northern portion of Section 28 have a general east-and-west strike, closely agreeing with the slaty cleavage; and the dips observed were all vertical.

The most easterly outcrop of slate in this belt occurs in the N. E. $\frac{1}{4}$ of the S. W. $\frac{1}{4}$ of Section 22, where it is again in exact contact with jasper of the northern belt, the strike in both rocks being about east and west and the dip vertical.

Immediately south of this main slate belt comes the principal jasper-area, in which are situated all the ore-bodies now worked at this end of the Vermilion range. In its broadest portion, on a north-and-south line through Montana shaft, this area has a width extending from the lake-shore at Stunz Bay to the south $\frac{1}{16}$ line of Section 27, or something over half a mile. From this line it extends west with hardly diminished width as far as No. 12 shaft. Thence westward the north and south boundaries converge, and come together probably in the N. E. $\frac{1}{4}$ of Section 29.

East of the line through Montana, the main area is divided into three parts by tongues of igneous rock, which reach west from the great mass lying in Sections 25 and 26. The most northern belt so separated is directly continuous with that part of the main area between Butte and Montana shafts and the lake shore at Stunz Bay, and extends east, past Siberia scram,

as far as the "burned 40's" in Section 23. It thus forms a continuous ridge two miles long east of Siberia, and is beautifully exposed. The trend of the ridge is further and further to the north in going east, so that the high hill, composed entirely of jasper, in the southern portion of Section 23 stands almost across the strike of the main slate area just described. In this distance the strike of the jasper bands follows the general direction of the ridge. Their true dip it is impossible to determine, owing to the extreme contortion of the rock, which is compressed into a multitude of sharp minor folds (Fig. 5), the axes of which follow the general trend. Besides the north and south compression indicated by the general direction of the axes, there has been enormous compression in an east and west direction also, with the result that the axes of the little folds show all angles of pitch, both to the east and west. The general effect of the close compression is to give one the impression that the dip of the rock as a whole is at an high angle, since on nearly horizontal outcrops the truncated edges of the steeply-dipping shanks are usually alone evident. But this effect is quite illusory; and it is probable that as a whole the rock is not highly inclined, although closely crumpled in detail.

Near the line between Sections 21 and 22 the strike of the jasper swings round to nearly east and west. Along this section line for nearly three-quarters of a mile to the north, the jasper striking nearly east and west, and dipping vertically, is beautifully exposed, penetrated in the most intricate way by sheets and dikes of quartz-porphyry and greenstone, as has already been described. At the northern limit of this cut area we find the strike nearly northwest, and the jasper dipping to the northeast.

These northern outcrops are the eastern termination of the most northern jasper-belt. To the west, jaspers are exposed on the mainland and neighboring islands, penetrated by porphyry, and much contorted, but plainly dipping towards the north and northeast. The axes of the minor folds have a general easterly pitch. These outcrops are found at short intervals as far as the north shore of Stunz Bay, where exposures are particularly good, and where a contact with the main slate belt has already been mentioned. West of Stunz Bay this belt is thrown out

into the lake by the great laccolite of porphyry, and it is only in Section 20 that it is again exposed in the headland east of Swede Bay, where, as already described, it dips to the north, and overlies the northern limb of the main slate anticline. This belt, exposed at intervals from Swede Bay on the west to Section 24 on the east, where it joins the northern prong from the main jasper-area of Soudan hill, constitutes the second jasper-area.

The middle finger from the Soudan hill area, is a very acute triangle, extending due east from No. 1 shaft for about one-quarter of a mile. On the north, it is separated from the northern finger by a wedge of intrusives, in which both greenstones and porphyries are represented, and which comes in from the eastern area, and seems to terminate near Montana shaft. On the south it is divided from the southern prong by the great mass of greenstones and greenstone schists which reaches its western limit near No. 7 shaft. It ends in a point, and the igneous masses north and south of it come together.

This middle finger shows the most extreme contortion, so that its structure cannot be precisely determined; but there is reason to conclude from the direction and pitch of the little folds that it is a closely compressed and probably shallow syncline, and that the igneous masses which nearly surround it are really continuous beneath it also.

The south finger is shown only by drill-holes at its western end. It leaves the main mass a little west of the Alaska crusher, and is separated from it by the rapidly-narrowing end of the immense greenstone sheet which is partly exposed on the surface in the railroad cut between Alaska and No. 1 shafts. For nearly a quarter of a mile southeast of Alaska there are neither drill-holes nor outcrops to prove the eastward extension of this southern finger, and the evidence for its existence rests on the ridge which leaves Soudan hill at the crusher and runs south of east towards Jasper Peak in Section 35. On this ridge, west of the southeast corner of Section 27, jasper again outcrops, greatly contorted, but showing an easterly pitch with a northerly dip. The same rock outcrops at short intervals on the flat top of the ridge as far as Jasper Peak, where magnificent exposures are continuous for half a mile. On the top of the peak, excellent structural observations are obtainable, and

the strike is seen to run in a general east-and-west direction, and the dip to be to the north at about 70°. Jasper continues east of the peak, still striking east and west, and standing either vertical or dipping north at high angles.

The main area of jasper, therefore, begins in the west as a unit, which rapidly broadens in going east to a maximum of half a mile. It then sends out three diverging fingers towards the east. The northern finger extends continuously to the northeast for two miles, forming the northern boundary of the great greenstone-area, and finally joins the second jasper-area on the line between Sections 23 and 24. The middle finger diminishes to a point in the main area of intrusives, under which it dips to the north.

The third area of jasper lies detached from the other two, and outcrops in two east-and-west hills north of the village of Tower. The eastern of these hills is generally known as Lee hill, from the name of the mine formerly worked on its eastern end. The western we may call Tower hill. These two hills both have their long axes east and west, but stand *en échelon*, the top of Tower hill lying half a mile north as well as nearly half a mile west of Lee hill. Both are composed mainly of jasper, with which, on Lee hill particularly, a large amount of sericite- and chlorite-schist is associated. On Lee hill the strike of the jasper, where observable at all, is mainly east and west, and the dip vertical; but on the northwest side the strike changes to a general northwesterly direction, with many contortions, and the rock may be followed across the intervening saddle, a distance of half a mile, to Tower hill, without important break.

West of the middle point of this saddle, a tongue of black carbonaceous slate comes in from the west, flanked both north and south by jasper, and tapers to a point in the jasper in the N.W. ¼ of the N.E. ¼ of Section 32. At the eastern extremity the jasper may be followed completely around the slate tongue, which is seen to be a closely compressed anticlinal fold, the axis of which pitches east at an angle of about 60°. The jasper adjoining the slate on the north dips high towards the north. The slate tongue rapidly widens in going west, and in about a quarter of a mile, when it disappears underneath the drift, the southern side strikes southwest with a vertical dip, while the

northern strikes northwest, dipping northeast about 65°. The jasper south of the western slate-exposures turns almost north and south, and dips towards the east at an angle of about 70°, and forms the western rim of an easterly pitching canoe, consecutive to the anticlinal on the north. The consecutive anticlinal south of this syncline is shown by exposures of slate farther east, which, however, are so closely compressed that they do not themselves show the anticlinal structure. But they also are cut off to the east by contorted jasper, which mantles completely around them, with high easterly pitch. South of this second slate anticline, and west of the point where the slate disappears beneath the jasper, the jasper turns again to the south and dips east, forming the prow of a second consecutive syncline.

These four little folds, the two slate anticlines and the two jasper synclines, all have the east-and-west axial direction, but stand *en échelon* from north to south, precisely like the larger topographical arrangement of Tower and Lee hills. To the west, in the line of strike of the axes of these folds, the ground is low, swampy, and drift-covered, and no outcrops are visible until the lake-shore is reached, which similarly consists of drift-materials, except at the single locality already described, where a coarse fragmental rock is exposed at the water's edge in the S.E. $\frac{1}{4}$ of the S.W. $\frac{1}{4}$ of the S.E. $\frac{1}{4}$ of Section 30. This rock strikes N. 45° W., and dips northeast about 60°, and from its position belongs to the northern limb of the first slate anticline, and represents a lower horizon than any rock there exposed. In this little area we have, then, the relative positions of the slates and jasper distinctly shown, and the slate is unmistakably the underlying rock.

On Tower hill, the jasper is beautifully exposed, and everywhere seems to stand vertical or to incline slightly towards the north. The strikes, while contorted, are mainly towards the north of west on the western slope of the hill, which falls abruptly into flat and swampy ground, in which no rocks outcrop. On the longer northern face, the jasper strikes more nearly east and west, or even to the north of east, and is also cut off (as well as on the east) by the flat, drift-covered and swampy ground which covers the entire intervening space as far as Soudan hill. From this swampy area a few knobs of basic

igneous rocks protrude, but no sediments. Between one-half and three-quarters of a mile east of Tower hill, the second slate-belt is exposed, just south of the jasper-area of Soudan hill. Its strike is nearly east and west, with a northerly dip; and there can be no doubt that it extends west into the covered ground north of Tower hill, and, with the usual intrusives, occupies substantially the whole area.

South of Lee and Tower hills is the low valley of East Two Rivers, without outcrops save of massive igneous rocks. The glacial drift, however, is abundantly supplied with angular fragments of slate. To the east, this hill gradually descends to the railroad track, and jasper striking east and west is occasionally exposed, or has been found in drilling, for the entire distance.

It seems very probable that a belt of slate extends all the way south of Lee hill in the Two Rivers valley, and finally joins the mass represented by the two little anticlines between Lee and Tower hills. We therefore regard the jasper belt of Lee and Tower hills as occupying a synclinal fold in which the two smaller anticlines are subordinate features; the northern, however, being important enough to find topographical expression in the saddle separating the two hills. That this greater syncline pitches east is probable, not only because that is the direction of pitch in the subordinate folds, but also because the jasper-area widens rapidly in going east from its western limit. East of the railway to Soudan the extension of this belt is not known. Jasper, penetrated by greenstone and sericite-schist, occurs near the schoolhouse in the village of Soudan. It is not impossible that the northern limb of this synclinal fold may connect directly with the southern prong from the main jasper-area of Soudan hill, somewhere south of Alaska shaft, so mantling over the anticline between the two hills, in which slate only is represented farther west. But no facts bearing directly on this question, except the occurrence near the schoolhouse, just mentioned, are at hand; and such connection is not indicated upon the map.

From the surface-distribution of the two sedimentary formations here outlined, it is evident that they are repeated several times in east and west belts, and that they are thrown into a series of pitching anticlinal and synclinal folds. On account of the close crumpling that has taken place from east to west,

as well as from north to south, structural observations have not always a definite value; but, it is nevertheless clearly shown that the slate underlies the jasper. The relative position of these two formations is important; for upon it depends the interpretation of the structure, and therefore, the evidence is summarized here.

1. The main slate-area has in itself an unmistakable anticlinal structure; both north and south of this area the jasper immediately succeeds the slates, and the jasper on the north dips away from the slate at relatively low angles. The two jasper-belts join at the east, completely cutting out the surface-extension of the slates in that direction. We can explain the junction of the jasper-belts in two ways only: It indicates either a synclinal with a westerly pitch or an anticlinal with an easterly pitch. But the minor subordinate folds in the northern belt of jasper have an easterly pitch; the northern jasper-belt, on the whole, dips away from, and not towards, the southern jasper-belt; and, finally, the included slate-belt is visibly an anticline. There seems, then, no loop-hole of escape from the conclusion that the two belts of jasper correspondingly succeed the slates on the north and south limbs of the anticline, and that the slate is the underlying rock.

2. The structure and relations of the little slate tongues and surrounding jasper west of Lee hill show, beyond doubt, that the slate occurs here also in anticlinal folds pitching east, and that it underlies the jasper. In these little folds the larger structure of the main slate-area is repeated in miniature.

3. Outside and north of the area mapped other east-and-west belts of slate and jasper occur within a few miles, which we have not studied in sufficient detail to justify mapping. The most northern—and, therefore, presumably, the oldest—of these belts is a slate belt, separated from the older crystalline schists by an area of basic igneous rock, which is probably intrusive between them.

4. Finally, at Ely, 22 miles to the east, we have similar slates overlain by similar jaspers. Upon the occurrence at Ely we do not insist very strongly, since we cannot yet show the continuity of the Vermilion Lake slates and jaspers with the similar Ely rocks; but that they are continuous we do not doubt.

We have, then, for the most northern structural feature an anticlinal fold, along the axis of which slates only are exposed for $3\frac{1}{2}$ miles. The slates are overlain on the north by northerly-dipping jasper. On the south they are also overlain by jasper, which occupies the consecutive and complex syncline of Soudan hill. South of this jasper syncline we find northerly-dipping slates again, which represent the northern limb of a second anticline, the southern limb of which is not exposed. South, again, we find the jasper of Lee and Tower hills, which appears to form the southern and western edges of a complex easterly-pitching syncline, the northern rim of which may be represented in the same mass, in which case the syncline is closely-compressed or may be drift-covered. Finally, all these folds pitch in the same direction towards the east.

VI.—The Origin of the Conglomerate-Breccias.

Returning now to the conglomerate-breccias, our views as to the origin of this structure will be clear when we take into consideration the physical characters and relative positions of the three kinds of rocks which were present in this series at the time when the great lateral compression, which has strongly folded the whole series, began. The sedimentary formations were then essentially horizontal, except where they had been dislocated and uplifted by the larger intrusions of igneous rock. Both slates and jaspers were thinly bedded, the sedimentary laminæ of the slates being still closer together than those of the jasper. One above another in the sediments were wide-spread sheets of igneous rocks, quartz-porphyry and greenstone, some as thin as the sedimentary laminæ, others many hundreds of feet in thickness. When this series was compressed, the thinly-leaved slates and jaspers could bend, for they were traversed by innumerable parallel bedding-surfaces along which fracture was more easy than in any other direction. Their structure was exactly like that of thin sheets of paper piled one on top of another. One little sheet could readily move over its neighbor, and so the mechanical conditions of bending, by which individual layers are compressed or pulled out in proportion to their distance from the neutral surface, could be satisfied. The intruded sheets of porphyry had no such surfaces of division running through them, but

were entirely massive, made up of particles firmly bound in a homogeneous matrix, and were equally strong and unyielding in all directions. Such material constituted a large part of the total thickness that was subjected to the compression, and in the greater sheets and laccolites accommodation by close folding would be clearly possible until the elastic limit had been passed. Within this limit, no doubt, gentle bowing was effected, and the main anticlinal and synclinal axes were determined. The result of the elastic resistance of these great interbedded sheets was to hold the general folding back, and for a time to prevent the close compression of the larger arches and troughs. During this period the sediments were squeezed between the unyielding masses and against their inclined boundaries, giving way in the little buckles that we see in the minor contortions. When the ultimate strength of the massive rocks was finally passed, then planes of actual fracture were developed in them, along which the relative movements which are necessary in folding could take place, and thenceforward they would offer only frictional resistance to the lateral stresses. In a word, in the slates and jaspers original minima of cohesion existed in one direction at close intervals; in the porphyries there were no such original minima. Beyond gentle bowing, and probably the outlining of the present anticlinal and synclinal axes, the porphyries could not yield until such minima were developed; that is, until they broke. Up to their breaking-point they would, therefore, hold back the general folding of the series. After breaking they would behave *en masse* precisely like sedimentary rocks occupying the same space, but with internal accommodations determined by the number and directions of the actual fractures. Considered as interbedded masses confined between upper and lower boundaries, the boundaries would follow the curves of the enclosing sediments, just as if sedimentary, and not igneous, rocks were confined between them. But the internal adjustments were altogether different from those that would have taken place in sedimentary beds, and depended upon the quite different distribution of the induced minima of cohesion, viz., the actual fractures.

We have seen that in fact these rocks show at least two principal directions of minimum cohesion, intersecting each other, and that by them the porphyries are divided into blocks. The

porphyries, after breaking, were thus practically masses of close fitting rubble, held together under great pressure, in which the total internal readjustment in folding was brought about by movements of each block relatively to those surrounding it, against great frictional resistance. The wearing of the contact surfaces, and the rounding of the angles of the blocks would inevitably result in the production of the conglomerate-breccia structure in greater or less perfection, according to the sharpness of the folding. It is noteworthy that the most perfect examples of this structure occur all along the axis of the northern anticline, particularly on the north shore of Stunz Bay, and in the " burnt 40's." They occur where we should expect them, on the lines where the folding has been most severe. Similarly, the great greenstone-area, in Sections 25 and 26, which lies along the eastern prolongation of the Soudan synclinal, is frequently characterized by this structure, although the development of the pebble-forms is not so perfect as in the more closely compressed northern anticlinal. This greenstone sheet, which is apparently the thickest and most widespread of the intrusive masses, occupies the surface where the Soudan syncline is widest, and therefore least closely flexed. It dies out as an integral mass at the north and south line through Montana shaft, beyond which to the west it penetrates the jasper in a great number of separate tongues. The coincidence of the main mass of greenstone with the sudden broadening of the Soudan syncline is probably not accidental, and is perhaps to be explained by the consideration that this great laccolite offered, to the end, more or less effectual resistance to folding.

The development of the conglomerate-breccias is thus seen to depend upon purely mechanical causes, and to be a necessary result of the division of the eruptive rocks into blocks. It is quite conceivable that these divisions might have taken place in the most irregular way, and given an end-product in no way differing from that which we see. There would be no difficulty in supposing that the resulting cores might have been turned by the continued pressure, and brought into line within the selvage produced by the rubbing of one block against another. But the initial and intermediate stages would then have been wholly different; and from the essentially regular distribution of the main systems of partings which we now see in these

stages, it is evident that they were not irregularly developed, but were produced in accordance with some definite law. It must be understood that the partings are not even and regular in a strict mathematical sense. In looking from a little distance at an exposed horizontal or vertical surface, the eye receives a strong impression of two parallel directions of intersection. A closer investigation, particularly where the structure is coarser, shows irregularity in detail, and considerable departures from parallelism. The main fact is that these two principal directions of parting point on opposite sides of the directions of cleavage and the axes of the greater folds, and that to this direction they bear a constant relation.

The reason for the development of these conjugate systems of parting is not wholly clear; but it seems very probable that the experiments of Daubrée* upon the rupture of glass plates under torsion, lately repeated by G. F. Becker† may throw light upon the phenomena here. It will be remembered that torsion, continued until the ultimate strength is passed, develops two intersecting systems of parallel fractures in the glass. These intersect nearly at right angles; and the axis of torsion bisects the angle between them.

It cannot be doubted that after the gentle bowings about several parallel anticlinal and synclinal axes began, the Vermilion series, with the heterogeneous physical characters of its separate members, and the irregular distribution, both vertical and horizontal, of the greater laccolites, was subjected to torsional strains, the axes of which closely agreed with the axes of folding. When we recall the observed relation between the folding axes, and the directions of parting, and the early stage at which the partings were developed, it seems very probable that torsion has played at least some part in producing them.

Our conclusions, therefore, with regard to the origin of this structure are these. The first step seems to have been the formation of intersecting sets of planes of fracture, dividing the originally massive rocks into roughly rhomboid blocks. Its further development depended on continued movement between these blocks under pressure, which resulted in enlarging the shearing-zones at the surfaces of contact, and rounding the

* *Études Synthétiques de Géologie Expérimentale,* 1879, p. 307, *et seq.*

† "The Torsional Theory of Joints," *Trans.*, xxiv., 130.

angles. The slate and jasper inclusions originally plucked off from the rocks which the porphyries and greenstones invaded, shared, of course, the subsequent history of their captors. The fact that the jasper inclusions are frequently rounded, while those of slate are not, is explained by the difference in the elasticity of the two rocks. The slate inclusions would readily yield and finally take a permanent set under the deforming forces, while the harder and more rigid jasper, in fragments of limited size and diverse orientation, would behave like the enclosing porphyry. The boundaries of the inclusions have generally been the surfaces along which rupture has taken place, although, as has already been said, jasper in a few instances has been found partly held in porphyry inclusions.

Similar structures in the aphanitic greenstones of the Marquette district have been described by the late Prof. G. H. Williams,* the origin of which he likewise ascribes to mechanical causes.

VII.—The Ore-Deposits.

The character of the iron-ores of the Vermilion range has been so well described by the Minnesota Survey that it is necessary here only to recall the fact that they are, in the main, very hard, massive, steel-blue hematites, for the most part without structure. Magnetites and specular slate-ores are very rare, and, indeed, do not occur at all in bodies of commercial importance.

The relations of the ore-bodies to the enclosing rocks, and their origin, are matters of greater interest, and as our conclusions on these points are at variance with those of the Minnesota Survey, it is necessary to present the facts at some length.

All the bodies now worked occur at two localities. The first is Soudan hill, where a group of closely-related deposits is worked as one mine, and the second is at Ely, some 22 miles to the east, where a single enormous deposit is the object of attack by the Chandler and the Pioneer Companies. In both the deposits of rich ore stand in close association with jasper and certain altered intrusives. These relations are of such a character that it seems to us they can be explained only by supposing that the ore-bodies are locally-enriched portions of the jas-

* *Bull.* 62, U. S. Geol. Survey.

per, due to replacement of the original siliceous or other constituents by ferric oxide. The places in which such replacement has affected large masses, and so produced workable deposits, are those in which igneous intrusives, both acid and basic, have been brought into peculiar attitudes with reference to the jasper by the extensive folding which the series has undergone. The time of the replacement appears clearly to have been subsequent to the folding.

It will be remembered that on Soudan hill, which runs nearly east and west, the jasper formation is most cut up by intrusives in the southeast portion. This is shown both by diamond-drilling and by exposures. Just south of No. 1 a drill-hole went through 900 feet of greenstone without encountering any jasper. The holes at Alaska shaft, about $\frac{1}{4}$ mile west, show a mass of greenstone about 500 feet thick, bounded by jasper to the north and south. West of this place the drill-holes do not show any such volume of greenstone in one mass, but they go through innumerable alternations of greenstone and jasper. At the extreme west end of the hill there is comparatively little greenstone, and that mainly in small dikes. The greenstone on the south side of the hill may be described as one connected mass, coming in from the eastward and penetrating the jasper toward the west, with innumerable branches, diminishing in size as they proceed from the main body. In places the jasper has been greatly shattered and penetrated in every direction by stringers and irregular masses of the greenstone, and wherever it is exposed, either on the surface or underground, the greenstone contains fragments of jasper.

The ore-bodies are on the sides and between the branches of this huge sheet, and are scattered over a considerable area, though the largest bodies are more or less in line. The longer axes of the bodies are, of course, always east-and-west, that being the direction of strike both of the jasper and of the intrusives. So great has been the replacement of jasper by ore that, for considerable distances, the jasper inclosed between two sheets of greenstone has produced an almost continuous ore-body nearly a mile in length. These bodies are, beginning at the east end, No. 1, Alaska, Nos. 7, 8 and 9. West of No. 9 are several other ore-bodies of irregular form but in the same general line. They are connected with a number of cross-dikes

in the jasper, but the great sheets which form the walls of the other ore-bodies do not extend west of No. 9.

The ore-bodies mentioned above are separated from each other mainly by small or large divisions of greenstone. These crossings are in most cases by no means of the nature of distinct dikes, but are merely ramifications of the greenstone laccolite. They occasionally cut the jasper at right-angles, and, indeed, at every angle, and they are almost always divided into a number of branches, where the greenstone has forced itself into every available crack in the jasper. These crossings invariably have a most pronounced effect on the ore-bodies which they cross. In general they dip toward the west (Fig. 8, Sections V. and VI.), and in every case within our knowledge the ore is cleaner and stronger on the upper side than on the lower.

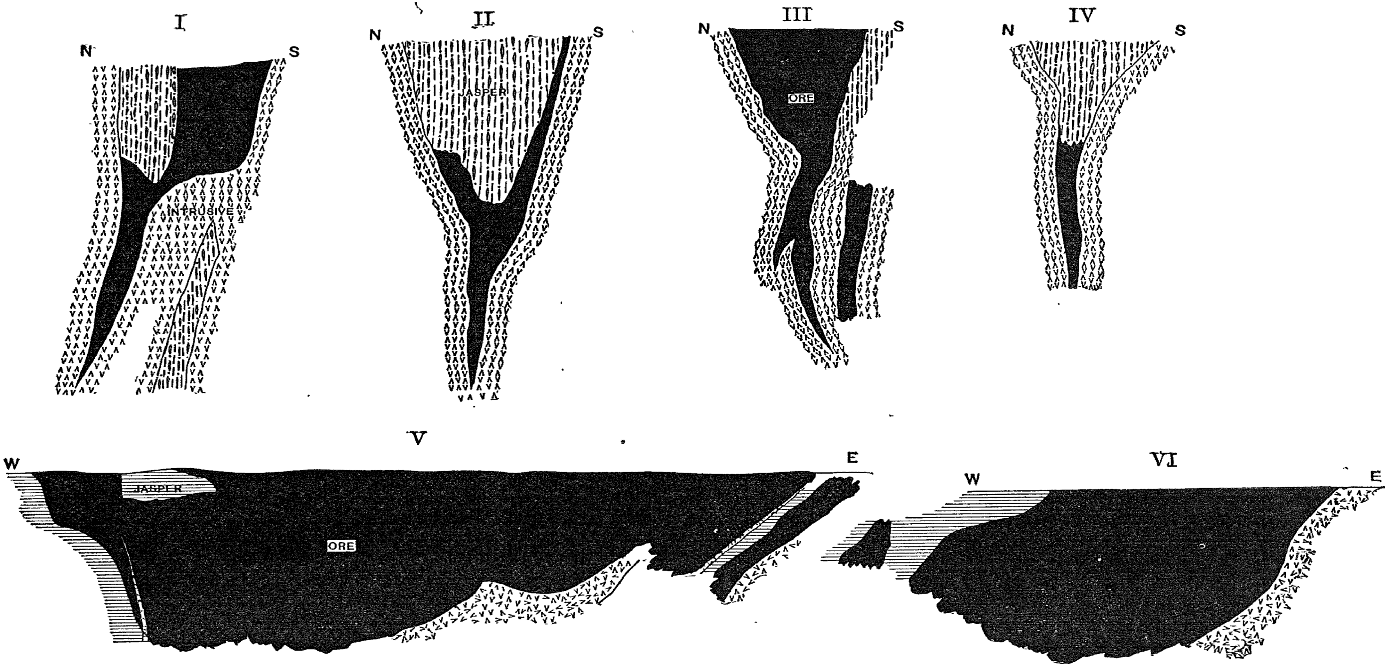
The ore-bodies worked by No. 1 and Alaska shafts are interrupted by no less than seven of these crossings, none of which are large enough to prevent all the bodies from being worked as one. The crossings all dip towards the west, and in doing so, of course, form, with one or the other of the walls, well-defined troughs pitching west. The ore most clearly follows the bottoms of these troughs; the upper portion of the ore so inclosed always contains more or less jasper. These bodies, so far as known, come to the surface at only two places—namely, at No. 1 and Alaska—being covered, in greater part, by a capping of lean white jasper from 100 to 200 feet deep, with fingers going down further.

Between Alaska and No. 7 is ground which has never been penetrated by the workings, but it contains one considerable body of ore between two large crossings.

The crossing which divides this ore-body from No. 7 is about 60 feet thick, and is the last which occurs between the great dikes that form the walls (Fig. 8, Section VI.). The trough formed by the walls and the crossing pitches westward—at the surface nearly vertically, but at a lessening angle as it descends; so that, at a depth of 650 feet, it seems to be nearly horizontal. In this trough occurs the great ore-body worked by Shafts 7 and 8, and there is reason to expect that it will connect at greater depth with No. 9. In cross-section this ore-body is about 600 feet by from 30 to 60 feet.

About 700 feet, on the surface, from the eastern end, the

FIG. 8.



SOUDAN HILL.—I. Cross-section near Shaft No. 6; II. Cross-section at Shaft No. 5; III. Cross section East of No. 5; IV. Cross-section at Montana Raise; V. Longitudinal Section through Shafts Nos. 3, 5, and 6; VI. Longitudinal Section of Ore-body No. 7 below the 5th Level. The portions in black include both ore now standing and ground already stoped. Scale of I.-IV., 1 inch = 200 feet; of V. and VI., 1 inch = 400 feet.

ore-body is interrupted by a mass of jasper which lies directly across the ore. The banding of the jasper and that of the ore are identical; and the former is evidently a mass of the original rock which has escaped replacement by the ore, though the reason for its escape is not plain. Another smaller body comes in just west of this jasper interruption, but, unlike all the bodies thus far described, it has no hanging-wall of greenstone. The dike which forms the north wall continues, as is shown by drill-holes, some distance farther west; but it has diverged so far to the north that, at the surface at least, replacement has not come quite up to it. This body of ore differs from all the others in this chain east of it in being a Bessemer ore. The ore in No. 9, another similar body lying on the same foot-wall some distance further west, is also a Bessemer ore.

It seems probable that the two bodies just described will join the great No. 8 ore-body in depth, and that they may be considered as fingers ascending along the foot-wall from the main body in the trough below.

South of this line of ore-bodies are a number of other small ones which have either not been worked or are too small to merit description.

The ore-bodies worked by shafts 2, 3, 5, 6, Montana and Butte, lie upon and between masses of intrusives that seem to be somewhat separated from those described above; but they have exactly the same form and general character. A great sheet, about 200 feet thick, forms the north or hanging-wall of all these deposits; and branches of much smaller size, which diverge from the main mass at small angles, form the various foot-walls.

The chief body has been worked by shafts 2, 3, 4, 5 and 6 and is a remarkably interesting body of ore (Fig. 8). The foot-wall is a dike about 15 feet thick, which leaves the main sheet some distance east of No. 2, and diverges steadily from it going west. At No. 2 the walls are 15 feet apart, at No. 3, 40 feet; at No. 4, 100 feet; at No. 5, 175 feet; at No. 6, 200 feet. The foot-wall is for the most part continuous; but it is nowhere a thick mass, and generally consists of a number of parallel thin sheets, overlapping one another, and not forming an absolutely solid and impervious mass like the great sheet to the north. West of No. 6 there appears an irregular mass of

greenstone cutting across from north to south in the manner of an irregular dike. This mass forms with the hanging-wall a trough, pitching east. The intersection of the foot-wall with the great northern sheet at the east end forms a trough, pitching west. The space enclosed by the three bounding masses of eruptives is an inverted irregular pyramid, the apex of which is 500 feet below the surface, and the base a triangle 1300 feet long by 200 in greatest breadth. The interior of this pyramid is jasper; most of the inclosing shell, ore.

The pyramid of jasper is 500 feet long, 100 wide and 200 deep, and lies against the hanging-wall. It is divided into two parts in the most curious and instructive manner by a small dike, of much later date than the igneous rocks of the walls. This dike dips east at an angle of 45°, and cuts the extremely contorted jasper nearly at right angles. The upper surface of this dike is followed by a body of ore which has replaced the jasper for several feet above the contact. This ore follows the dike quite across the jasper core to the hanging-wall.

In the foot-wall, the replacing solutions have penetrated through some of the overlapping sheets of greenstone and produced some irregular bodies of ore, connected with the main body and lying between the branchings of the foot-wall dikes.

West of these ore-bodies are the ones worked by shafts Montana and Butte, which have the same hanging-wall as those above described; but the arrangement of the ore is somewhat different. The hanging-wall is not properly so-called, for it is either vertical or dips slightly south, but it is called the "hanging" in contradistinction to the foot-wall which is a dike dipping north. Both walls distinctly cut across the jasper and by their junction form a V-shaped trough pitching west at an average angle of not over 10° (Fig. 8, Section IV.). Only in the bottom of this trough has the replacement been complete; and the upper portion is covered by a capping of jasper, the extreme contortion of which is in noticeable contrast to the regular and massive structure of the ore-body. In cross-section, this ore-body is about 50 feet wide at the top, diminishing to nothing at a depth of 200 feet below the capping. It is cut across by a number of irregular crossings or dikes which branch out from the walls.

On Lee hill a number of small bodies occur which have been

worked in the past, but are now abandoned for various reasons. The largest of these is the Lee, which gives its name to the hill. The long horizontal axis of this body runs east and west, which is the direction of strike of the jasper also. The south wall is lean uncolored jasper, and rock of the same character limits it to the west in the line of strike. On the north, greenish sericite-schists, known as "soap-rock," bound the ore, and likewise circle round it on the east and on the south, including between themselves and the ore-body the considerable thickness of lean jasper on the south wall. The line of côntact between these schists and the iron-bearing rock is well exposed, and is without doubt an igneous one. The schist cuts across the jasper bands and sends apophyses between them; and includes fragments of various sizes and shapes, from angular pieces to long narrow lenses and layers. The included jasper fragments are usually well banded, and often the directions of these bands largely depart from the direction of schistosity. The schists are chlorite- and sericite-schists, carrying many grains of quartz, pretty uniform in size, which show usually complete granulation. Under the microscope the fact that they are really sheared quartz-porphyries is evident.

The ore-body lies in the angle between the schists on the north and east, and the jasper on the west and south (Fig. 9). At the west end it passes into jasper, becoming first mixed, and the latter gradually predominating, leaving the ore finally as stringers in the jasper mass, the banding of which continues into the ore. The presence of the ore, it seems to us, cannot satisfactorily be explained otherwise than as due to a replacement of the jasper within this angle, which was in some way connected with the peculiar occurrence of porphyry.

The Chandler ore-body at Ely, which seems likely to prove one of the largest in the Lake Superior region, is for other reasons also of very great interest. It occurs in the bottom of a synclinorium, the axis of which runs a little north of east, and pitches toward the east. The trough has been closely folded and also slightly overturned, so that both walls are parallel, and dip north. About one-third of the distance from the south wall, a minor anticlinal fold divides the trough into two synclines. The southern pitches east at an angle of about 20°, while the pitch of the northern is much higher, or about

50°. The bottom of the southern syncline comes to the surface about 400 feet west of the northern; but on the 8th level, owing to the difference of pitch, these relations are reversed,

Fig. 9.

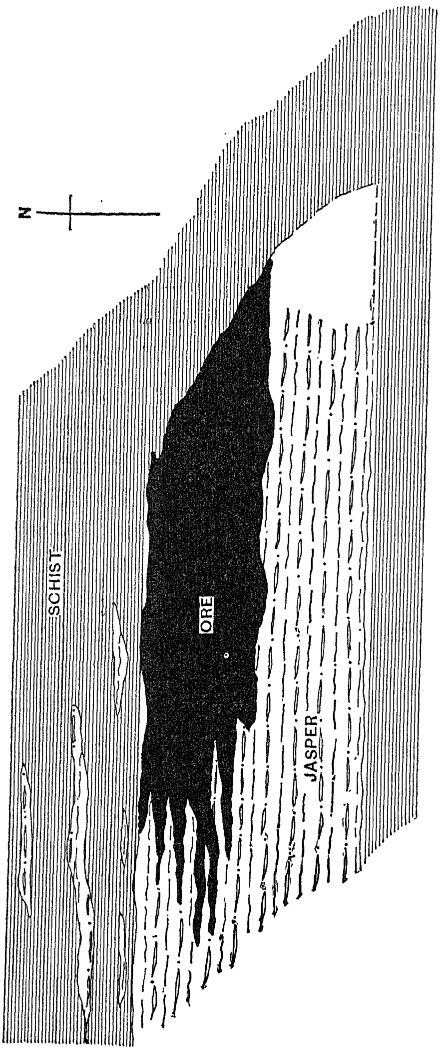


Diagram Showing in Plan the Relations of Jasper and Sericite Schist at the Lee Pit.

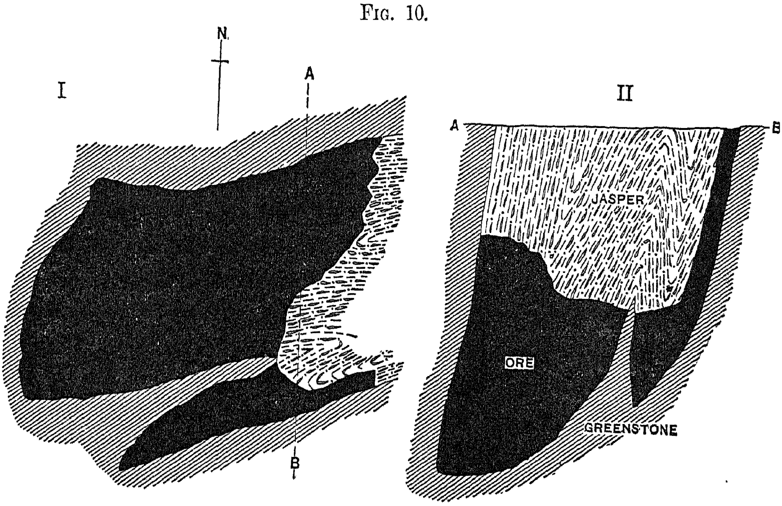
and the bottom of the northern fold is about 300 feet west of the southern. The same difference of pitch also accounts for the greater length on the surface of the south body. The upper and interior portion of this synclinorium is jasper, the bottom is ore (Fig. 10).

Underneath the ore, and on both walls, is a brecciated green-

stone, very fine-grained and indeed aphanitic, the relation of which to the jasper is unquestionably intrusive. Alteration has progressed very far in this rock; but thin sections still show lath-shaped plagioclase feldspars arranged in the manner characteristic of diabase. The other minerals are all secondary, and comprise calcite, chlorite, sericite, zoisite and quartz, which incloses needles of rutile.

From the greenstone walls innumerable small dikes penetrate into the ore and jasper, and near the surfaces of contact numer-

FIG. 10.



CHANDLER ORE-BODY.—I. Plan of the 8th Level; II. Cross-section. The indication of structure in the jaspers is diagrammatic. Scale, 1 inch = 400 feet.

ous fragments of the latter are included. North of the ore-body the greenstone is well exposed on the surface as far as Long Lake; and in this distance bands and layers of jasper are included in it, one of which has a great length and a considerable width. On the shore of the lake, some distance west of the mine, black slate is cut by the greenstone. This underlying greenstone, therefore, clearly appears to be an intruded sheet, the great mass of which entered the sedimentary series along the junction between the slate and jasper formations, which have the same relative position here as at Tower. Subsequently, all were folded together into their present attitude.

It has already been said that the greenstone is brecciated.

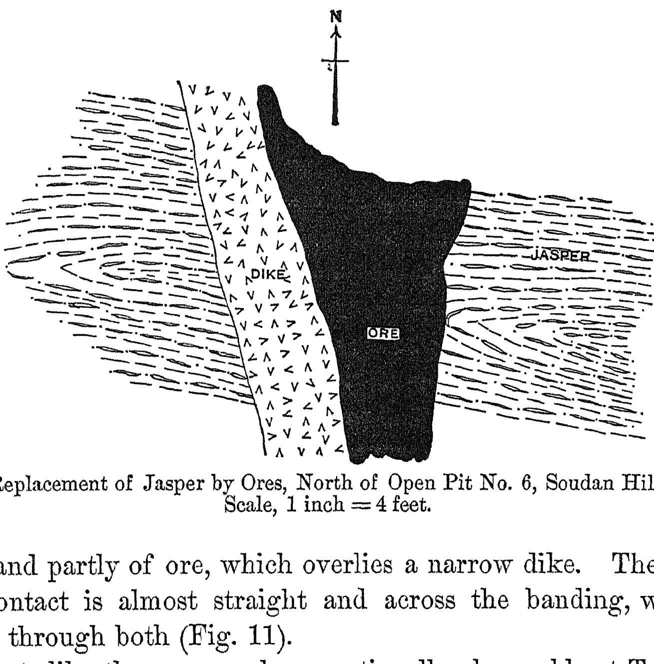
The ore and jasper above it have the same structure. The former, indeed, is hardly other than a rubble, the slightly coherent units of which are hard blue hematite. To this brecciation the ore owes its "soft" character, which, together with the similar structure in the jasper capping, has permitted the development of the perfectly adapted system of mining practised here. Such brecciation could result only mechanically from the action of great stresses, and, together with the form and attitude of the deposit, would in itself indicate the presence of a structural syncline. But the evidence for the syncline does not rest on these considerations alone. Three years ago the actual turn could be observed in the banding of the ore, at the west end of the south body, along the contact with the greenstone. At the present time the upper levels have caved along this line of contact, and the lower have not yet extended to it, so that now it is inaccessible for study.

The line of contact between the ore below and the jasper above, shown on the cross-section (Fig. 10), is a very irregular one but for most of the distance across the trough it is nearly horizontal, with an upward bend near the walls, slight near the hanging, but much more pronounced near the foot, with which, indeed, on the plane of the section, it is parallel. The line, as here drawn, is simply the commercial limit, above which the ore becomes too lean for the market. The actual passage from ore to jasper is gradual and not sharp. In this zone of mixed ore, fragments of the brecciated jasper are found in various stages of replacement. The units in the completely altered ore-breccias have precisely the same forms, occur in the same way, and have the same banding. It is impossible to doubt that the whole interior of the trough, within the greenstone boundaries, was originally a jasper breccia, like that which now partially remains, and that the ore-body owes its existence to the replacement of the silica of this jasper by ferric oxide.

We conclude, therefore, from our studies at Tower and Ely that the ore-bodies of this range were not formed contemporaneously with the inclosing rocks, but are metamorphic deposits which have been accumulated in certain favorable places by the replacement of one or all of the original constituents of the jasper, except the iron, by new ferric oxide. The fact of replacement in certain cases, as at Ely, can hardly be denied.

We have in our possession specimens from Tower also, which consist at one end of banded jasper and at the other end of ore. The structure of the jasper is perfectly preserved in the ore, and continues through both. The ore is, in fact, a pseudomorph after the jasper. The same thing can often be observed in the field on a larger scale. A beautiful example occurs north of No. 6 open-pit on Soudan hill. The iron-bearing rock is greatly contorted and thrown into small and closely-compressed folds. One of these consists partly of jas-

FIG. 11.



Replacement of Jasper by Ores, North of Open Pit No. 6, Soudan Hill. Scale, 1 inch = 4 feet.

per and partly of ore, which overlies a narrow dike. The line of contact is almost straight and across the banding, which runs through both (Fig. 11).

Facts like these are only exceptionally observable at Tower, where indeed the ore usually is entirely massive, and shows no structure whatever except jointing. But, broadly considered, the same general relations everywhere hold good. In the case of every known body, jasper forms at least one boundary in some part of it, under such circumstances that the bands, if continued, would run into the ore. This fact, taken in connection with the tortuous form of many of the bodies, seems to us quite inexplicable on any theory of contemporaneous deposition of jasper and rich ore. For such a theory would involve the extraordinary assumption that the conditions of sedi-

mentation or chemical precipitation were so radically different on opposite sides of an imaginary vertical plane in ocean water, as to permit the contemporaneous deposition or precipitation of nearly pure silica on one side and nearly pure ferric oxide on the other; and that such differences in conditions persisted long enough to permit the accumulation, in some cases, of 100 feet or more of material.

Furthermore, the massiveness and lack of structure in the Tower deposits show that they were formed after the folding took place. For if they had been in existence before the folding, it is impossible that they alone, of all the materials of the crust in this region, should bear no mark of it. The observed facts, therefore, both in their general bearing and in their minute details, lead to the view that the deposits of rich ore represent areas in the jasper formation in which an interchange of ferric oxide for silica has been effected subsequently to the general folding. These areas are so placed with reference to igneous rocks, and their association with these is so constant, that the conclusion that some causal relation exists between the two seems irresistible. The larger ore-bodies are found in places where, by folding (as at Ely), or by tilting and intersection combined (as on Soudan hill), the igneous rocks have been brought into the form of troughs opening upward, which inclose partially or completely large blocks of jasper. The ore occurs round the inner surfaces of contact, especially in the bottoms of the troughs where the sides meet, and away from these and above it gradually passes into the jasper.

These general relations between ore, jasper, and intrusives are precisely the same as those described by Prof. Van Hise for certain classes of deposits in the Marquette district, and we believe that his general conclusions as to the part played by the intrusives in localizing concentration are entirely applicable here, as indeed he has already maintained.*

He has shown that the igneous rocks, as compared with the jaspers, are relatively impervious to water, and that the occurrence of the ore on top of the relatively impervious material points to waters descending from the surface as the agents active in effecting the concentration, a view to which the com-

* *Am. Jour. Sci.*, 3d series, vol. xliii., pp. 130, 131, 1892.

pletely oxidized character of the product lends additional support. Such waters, bearing organic acids, slowly percolating downward through the iron-bearing member, would take iron into solution, and, because of the downward convergence of the bounding surfaces of each block, would be accumulated along the sides and in the bottoms of the troughs formed by the igneous walls. These, then, would be paths which all waters seeping down through each block would ultimately reach, and beyond which they could not readily pass, and so would become channels of maximum circulation. Along these paths the removal of silica would be facilitated by the alkalies furnished by the alteration of the adjoining igneous rocks. The surfaces of contact between jasper and intrusives being in most cases surfaces of slight dislocation during the folding of the series, would be the readiest channels of descent, and down these channels waters sinking directly from the surface would carry free oxygen to the bottoms of the troughs. "The union of these two currents would precipitate the iron-oxide." *

Assay of Auriferous Ores and Gravels by Amalgamation and the Blow-Pipe.

BY R. W. LEONARD, M. C. SOC. C. E., KINGSTON, CANADA.

(Atlanta Meeting, October, 1895.)

A METHOD of accurately determining the value of gold-bearing ores and gravels with a compact field-outfit is much desired by prospectors, especially when the work lies in a district far removed from railways or good wagon-roads.

We are all familiar with the fire-assay of gold-ores, and fully recognize its accuracy and value; but only those who have tried to determine the value of a gold-bearing vein as a commercial venture, from fire-assays of selected samples, know how misleading is the information thus obtained, however carefully and honestly the samples may be selected.

The reasons are that for the fire-assay a very small portion of ore is treated (rarely more than six assay-tons, or a little over one-third of a pound), and a minute particle of the precious

* *Am. Jour. Sci.*, vol. xliii., p. 127.

metal, accidentally included or excluded in taking the sample, will show a very large difference in the resulting value of the ore. The fire-assay also gives the whole value in gold and silver in the ore, without indicating what proportion of the precious metal is free-milling, and how much must be recovered by other chemical processes from the concentrates or be lost in the tailings. To avoid being misled by these defects, a cautious engineer will recommend a mill-test of several tons of ore, the cost of which is, in many instances, considerable.

It was with the purpose of treating a much larger sample of ore than can be conveniently treated by fire-assay that I returned to the use of mercury. By this means I obtained a fairly reliable estimate of the value of the ore with compact and portable apparatus, and, at the same time, the free-milling contents, as well as the value in the concentrates. By some modifications of the system of treating the amalgam, all the information resulting from a mill-test may be obtained with considerable accuracy.

I present this record of the somewhat meager results of my experiments, in the hope that it will be of interest to the profession.

The operation described is such as would be carried out by an engineer sent to report on the value of ore in sight and the method which should be adopted to treat it.

The ore is sampled by one of the approved methods, and a sample (of say two tons) is broken on a close floor or level rock to a small size and quartered. The selected quarter is then broken to a smaller size and again quartered, until a sample of several pounds is obtained, which is then crushed in a mortar to pass through a sieve of the desired fineness.

A sample of the crushed ore is then carefully weighed out, taking from 10 to 100 assay-tons, or a quantity deemed sufficient to give an average value of the whole ore and a button large enough to be weighed. (A good balance will weigh accurately to one-tenth milligramme, and one milligramme of pure gold from the ten assay-tons would mean a value of $2.06 per ton.) This selected sample is rubbed, in lots of three or four assay-tons each, in a mortar with a small amount of distilled and cleaned mercury and water. After treating one portion long enough to insure complete amalgamation of the free gold, the

mercury is panned out for use with the next portion, and the ore is saved for further careful panning to recover the last particle of mercury and the concentrates for weighing and further treatment. When the whole weighed sample has been amalgamated it is carefully panned over for the remaining mercury and the concentrates. The mercury is now placed in a small cast-iron retort (bored smooth to a small diameter in the bottom) and carefully retorted. When all the mercury has been distilled off, a little test-lead is poured into the hot retort and melted, to absorb any bullion that may have been contained in the mercury. By this method there is no loss whatever of the precious metals, and the smallest particle of gold collected in the mercury can now be recovered from the lead by cupelling—preferably with the blow-pipe. The resulting button is weighed, parted and weighed again, and the value of the ore in free-milling gold is calculated.

The concentrates are now dried and weighed and the percentage is calculated. As the gold is very evenly disseminated in a fine state of subdivision in the concentrates, an assay of a small portion is sufficient to determine its value.

If the assayer is expert in the use of the blow-pipe, he can without difficulty make an accurate assay of the value of the concentrates, if they are sufficiently rich to warrant saving. The process is simply that described in Cornwall's *Blowpipe Analysis*, and therefore need not be repeated here. I prefer, however, the use of the paraffin-lamp to the Berzelius oil-lamp, on account of its simplicity and the fact that the larger flame of the paraffin-lamp will permit the treatment of three centners of ore (or even more) at each fusion.

In measuring the resulting button (for it will be too small to weigh), it is necessary to use a scale which is known to be accurate. Appended to this paper are some results obtained by measuring a number of silver buttons, and afterwards weighing them, which show the degree of accuracy attainable in this work by using a carefully-made scale. In this case the buttons were measured on a Plattner's scale manufactured by Carl Osterland, of Freiberg, Saxony.

The following are a few results of assays which I carried out by the method above described, compared with the results of fire-assays and of mill-tests of considerable quantities:

1. Fire-assay of 1 assay-ton of quartz with mispickel from Marmora district: value, 7.00 ounces per ton.

	Ounces per Ton.
By amalgamation: value, ...	2.50
Blow-pipe assay of 54.4 per cent. concentrates (value, 6.50 ounces per ton): value, ...	3.48
	5.98

2. Fire-assay, quartz with mispickel: value, 0.10 ounce per ton.

	Ounces per Ton.
Amalgamation (12 assay-tons): value, ...	0.14
Blow-pipe assay of 36 per cent. concentrates: value, . .	0.03
	0.17

3. Fire-assay (6 assay-tons quartz with calcite) gave 0.00 ounce per ton. Mill-test of 4 tons gave small button, suspected to belong to the plates from previous runs. The amalgamation of 51.5 assay-tons gave a value of 6 cents per ton. No concentrates.

4. Three samples of 20 assay-tons each of mineralized quartz, gave by amalgamation only a trace of gold and 1.5 per cent. of concentrates, showing by blow-pipe-assay a value of 0.00.

Fire-assay of 6 assay-tons of ore gave only a trace of gold.

" " 1 assay-ton of concentrates gave a trace of gold.

No mill-tests made.

5. Quartz, with mispickel and pyrite.

	Ounces of Gold per Ton.	Ounces of Silver per Ton.
Fire-assay (6 assay-tons): value, ...	0.091	0.003

	Ounces of Gold per Ton.	Ounces of Silver per Ton.
Mill-test of 2000 pounds: value, ...	0.251	0.030
3.275 per cent. concentrates: value, 3.51 ounces of gold and 0.013 ounce of silver, or, for the original ore, ...	0.115	0.0004
Tailings: value, ...	0.021	0.001
Total value of ore, ...	0.387	0.0314

	Ounces of Gold per Ton.	Ounces of Silver per Ton.
Amalgamation (16 assay-tons): value, .	0.351	0.039
3.57 per cent. concentrates assayed by blow-pipe (2.65 grammes), ...	0.106	0.012
Total value by blow-pipe method, . .	0.457	0.051

I regret that I am not able, as yet, to give the results of as many or as complete experiments as I would desire. The few given above, however, are sufficient to indicate the accuracy of this system of assaying auriferous ores. The ap-

plication of the system (of amalgamating and afterwards recovering the value by cupelling the test-lead in the manner above described) to auriferous gravels is too apparent to require further explanation. It is possible that some hydraulic mining engineers may use this system, although I have never heard of it. The results of Assay No. 3 are sufficient to show its value for this purpose.

The value of the system is measured by its accuracy, the amount of information it furnishes as to the character of the ore and the method to be adopted in working it, and the facility with which the prospector or mining engineer can carry out the work in inaccessible districts with a plant easily carried in his travelling-bag.

I beg to acknowledge, with thanks, the kindness of Dr. Goodwin and Professor Nicoll, of the Kingston School of Mines, for the results of the mill-tests and some of the fire-assays in connection with them.

The mill-tests were carried out with a small three-stamp testing-mill at the Kingston School of Mines.

The following tables, kindly contributed by Captain J. B. Cochrane, instructor in chemistry at the Royal Military College of Canada, give the results of some experiments conducted by him to ascertain the accuracy with which small silver buttons can be weighed by measurement on a Plattner's scale. It will be observed that the average weights by measurement exceed the actual weights by about 2 per cent. only.

First Set.

	Scale Measurement.	Weight on Assay Balance.
	Milligrammes.	Milligrammes.
1	1.47	Not weighed.
2	.97	" "
3	3.07	3.20
4	1.34	Not weighed.
5	1.78	" "
6	3.01	3.00
7	2.87	2.84
8	.64	.64
9	.61	Not weighed.
Total	15.76	15.40

Total error, + 0.36 milligramme.

SECOND SET.

	Scale Measurement.	Weight on Assay Balance.
	Milligrammes.	Milligrammes.
1	2.47	Not weighed individually.
2	.51	
3	3.17	
4	.34	
5	1.18	
6	.57	
7	1.89	
8	2.16	
9	.89	
10	1.10	
Total	14.28	13.93

Total error, + 0.35 milligramme.

THIRD SET.

	Scale Measurement.	Weight on Assay Balance.
	Milligrammes.	Milligrammes.
1	1.91	Not weighed individually.
2	2.59	
3	.25	
4	.37	
5	1.84	
6	2.30	
7	1.96	
8	1.71	
9	3.27	
10	.91	
11	1.09	
12	2.01	
13	1.91	
Total	22.12	21.72

Total error, + 0.40 milligramme.

An Improved Form of Protractor for Mapping Mine-Surveys.

BY W. S. AYRES, HAZLETON, PA.

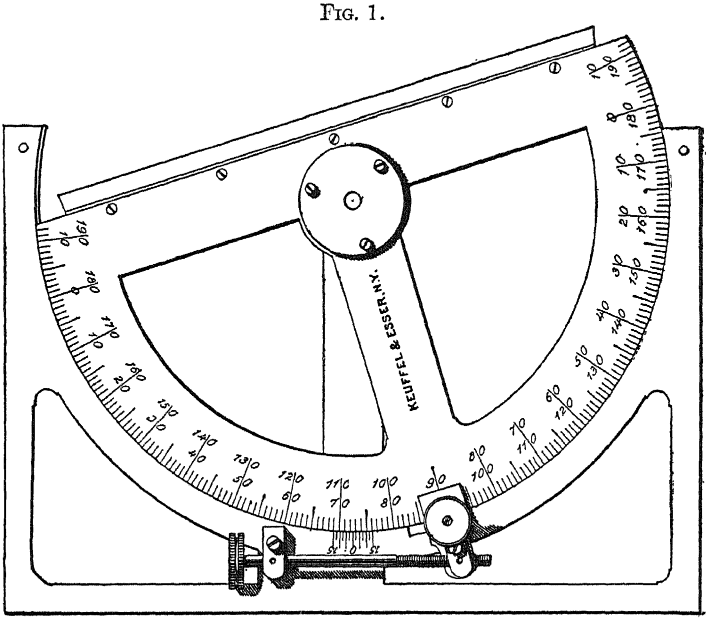
(Atlanta Meeting, October, 1895.)

THE protractor here to be described embodies several important improvements that have been suggested by the use of other protractors and by observing the errors occurring in practice with them. It is more particularly an improvement

on the protractor designed by Crozet of the U. S. Engineer Corps, an illustration of which, kindly furnished by the manufacturers, Messrs. Keuffel and Esser, of New York City, is given in Fig. 1.

This old style is divided into $\frac{1}{4}$-degree graduations, with a 15-minute vernier, reading to 1 minute. It was a mechanical impossibility to graduate a protractor of this form into the

Fig. 1.



Crozet Protractor. Old Style.
Eight inches, divided to one-fourth degree, with vernier reading to one minute.

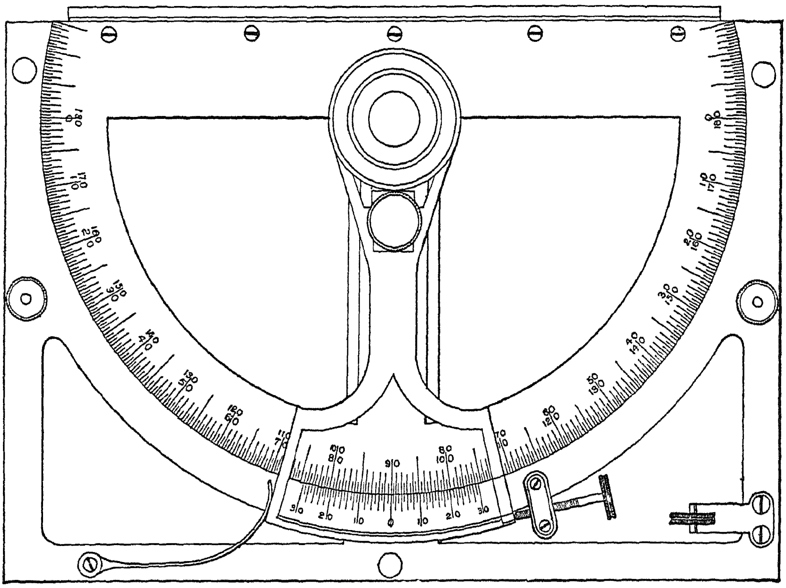
same divisions as a transit, viz.: into $\frac{1}{2}$-degrees with a 30-minute vernier reading to 1 minute, owing to the fact that the old style of clamp covered up a portion of the graduations when the angle to be turned off approached the perpendicular to the base or the zero on either side. The graduation into $\frac{1}{2}$-degrees has been made possible, however, by the device shown in Fig. 2, which illustrates the improved form.

In this new arrangement the clamp is placed at the center and away from the graduations, leaving the vernier entirely open to unobstructed view. It is operated by the screw near the center which in turn operates a wedge that presses a flexible piece against the hub of the arc, thus clamping it. The vernier

is adjusted to the exact minute by the tangent-screw, operating against a spring, as shown in the figure.

The advantage of having the graduations of the protractor the same as those of the transit will be apparent to every engineer. The frequent errors made, due to the employment of different graduations, are avoided. There is no mistaking a $\frac{1}{4}$- for a $\frac{1}{2}$-degree in plotting, as is often done in practice, when the graduations of protractor and transit are not alike.

FIG. 2.



The Ayres-Crozet Protractor.
Eight inches, divided to one-half degree, with vernier reading to one minute.

It has been found necessary also to place an ivory support immediately at the center of the arc, in addition to the three formerly used in the frame. This necessity is made plain by noting that, when the arc is revolved until the ruling edge closely approaches a position perpendicular to the base of the frame, the weight of the arc is nearly all upon the center. This center is fixed in position by an arm attached to the frame and extending under the arc as shown in the figure. Without this support of ivory the arm would spring, and the plane of the arc would not coincide with the plane of the vernier when the ruling edge approached this perpendicular position, thus causing difficulty and uncertainty in the adjusting of the vernier.

A thumb-movement has also been provided, consisting of a milled wheel and flexible spring attached to the frame, as shown in the lower right-hand corner of the figure, by which, from a simple pressure of the thumb bringing the wheel down upon the paper, and by a slight rolling movement the protractor is moved along the straight edge until its ruling edge is brought directly, and without jar or slip, upon the point from which the line is to be drawn.

Having the graduations in ½-degrees, the divisions on the vernier are very clear, and the reading from the transit can be repeated on the protractor exactly in mechanical form as well as exactly to the minute. This similarity to the transit, together with the support at the center, which makes it impossible for any part of the metal of the protractor to soil the paper, and the thumb-movement described, have been found after several months of use to lighten greatly the laborious work of plotting surveys, insuring at the same time greater accuracy and greater expedition.

The improved protractor shown in Fig. 2 is made by Messrs. Frank C. Knight & Co., of Philadelphia, from drawings furnished by the writer.

Specifications for Steel Rails of Heavy Sections Manufactured West of the Alleghenies.

BY ROBERT W. HUNT, CHICAGO. ILL.

(Atlanta Meeting, October, 1895.)

IN 1888 the writer had the honor of submitting to the Institute a paper on "Steel Rails and Specifications for their Manufacture."* In his judgment the specifications were sufficient for "that day and generation." In that paper I said:

> "The first duty falls upon the railroad engineer in designing his section. If that is bad, the steel-maker will be heavily handicapped in trying to furnish a satisfactory rail. I suppose it is hopeless to expect the adoption of standard sections."

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* *Trans.*, xvii., 226.

That which I dared not hope has practically come to pass. In 1885 the American Society of Civil Engineers appointed a committee to consider "The Proper Relations to Each Other of the Sections of Railway Wheels and Rails." Their labors were supplemented by those of a later committee of the same Society, appointed in 1890, and directed to submit a report on a series of rail-sections, ranging from 40 to 100 pounds per yard, and varying by 5 pounds.

It fell to my lot to be a member of the last committee, and its Secretary, during the final and greater part of its labors. I mention this as evidencing my familiarity with the honest and careful efforts made to obtain the views of the leading railroad engineers of the country, to harmonize differences and to design a series of rail-sections which would be in accord with their experiences and generally acceptable. At the same time steel-makers were consulted, so that the proposed sections might not present special difficulties in manufacture.

Three years of faithful work on the part of the committee were required before it was able to make a full report to the Society. In August, 1893, the report was accepted and the committee was discharged.

While the Constitution of the American Society of Civil Engineers prevented that society from officially adopting as its own the rail-sections recommended by the committee, they have been popularly so regarded and called; and, what is better, they have been largely adopted already by the railroads of the country, and promise soon to be absolutely the standard American sections.

In my former paper I declared my conviction that the disappointing wear so often obtained up to that time from the heavier-sectioned rails had been "largely the fault of the sections themselves." As I believe the standard sections just named are good ones, it is my opinion that, if they fail to give satisfactory service, we must look in other directions for the cause.

The years of continued experience since 1888 have made me all the more positive in my estimate of the supreme importance and influence of the physical and mechanical treatment of the metal while being converted into steel and cast into ingots, and of the steel while being made into rails.

Since the presentation of my former paper, many changes in the routine of manufacture have been generally adopted by American rail-makers. "Hot heats" in the converter are strenuously avoided; greater care is exercised in teeming, and as to the character of the ingot-moulds used; it is exceptional to throw ingots on their sides while the interior metal is yet liquid; and vertical furnaces, or so-called gas-soaking-pits, have been adopted by the largest makers. By so handling the ingots and by the use of such furnaces, and by being liberal in cutting off the top-ends of the blooms, the danger of "piped" rails has been brought to the minimum.

My former statement, that "the character of the permanent way of the railroads of the United States is improving each year," still holds good, and the demand on the mills for well-finished rails is, accordingly, imperative. As a rule, this demand has been well met by the makers, and I know that the general finish of the rails delivered to the roads to-day is very greatly higher than it was seven years ago. A much stricter inspection has been insisted upon, and has borne its fruit.

The material interests of the country are just recovering from a period of great depression; the coming year promises to be one of large rail-purchases; and in many cases the roads will be compelled to replace comparatively light sections with heavier ones. The American Society rail-sections were designed to obtain as much work as possible from the rolls on the rail-heads. At the same time it is impossible to get as fine-grained steel in the head of an 80-pound rail as in a 56- or 60-pound one; and it must not be expected that a coarse, open-grained rail will wear as well as the finer one. To my mind this fact explains the whole "mystery" of the superior wear of the old light-sectioned rails, many of which, nevertheless, were of very bad chemical composition. Now, if we cannot obtain the resistance to wear from the fineness of structure due to work, we must seek it from hardness and soundness due to chemical composition. And as we increase the sections, we can with safety add to the hardness.

During the discussion* of Dr. C. B. Dudley's paper of 1881, I sought to defend silicon from some of the accusations brought

* *Trans.*, ix., 534.

against that element; and in a paper on "Bessemer Steels," read before the Franklin Institute of Philadelphia, January 25, 1889, I presented a large number of analyses of many kinds of steel to sustain the same position. Moreover, the makers of steel castings demonstrated long ago the value of silicon in obtaining soundness when added to the metal after conversion. In the specifications which Mr. P. H. Dudley prepared for the N. Y. C. & H. R. R. Co., he recognized the value of silicon; so have many European rail-makers. But while we are seeking to make our rails harder by chemical means, we must not lose sight of an element which, while hardening, also tends toward brittleness, namely, phosphorus, the *bête noire* of the steel-maker. Hence the increase of carbon-percentage must be governed by the amount of phosphorus present. Moreover, no matter how anxious we may be to obtain the best possible rails, commercial conditions must be recognized; and these are often controlled by geographical circumstances. It happens that the largest American deposit of Bessemer ore (which is also the cheapest) available to the makers of rails east of the Alleghanies, is low in phosphorus. The foreign ores which these makers would import have the same characteristic. Therefore, eastern railroads can obtain low-phosphorus, high-carbon rails without paying an extra price for them, and the rail-makers have no trouble in getting suitable ores. West of the Alleghany mountains the conditions are different. This difference has led me to prepare the specifications which I now have the honor to present.

The only practicable sources of ore-supply for the western mills are the Lake Superior districts; and, until lately, the available supply of ore, even moderately low in phosphorus, from these districts, was limited. For this reason the western standard of Bessemer pig-iron was 0.10 per cent. of phosphorus. Of course, steel rails made from such iron would contain quite 0.11 per cent. of phosphorus; and in many cases that element ran up to 0.12 per cent., and sometimes higher.

The later ore-developments in the Lake Superior region, particularly on the Mesabi range, have altered this situation, so that, next season, it will be perfectly practicable for the western rail-makers to keep the phosphorus in their rail-steel under 0.09 per cent. I know it will be said that even now there is not

enough low-phosphorus ore developed to permit the adoption of such a standard. I must take issue with this assertion. Ore-mixtures have been purchased heretofore, based on the expected total production of Bessemer steel by the purchasing works, and, as the metal came from the blast-furnace, it was made into soft steel for wire-rods, billets, tin-plate, bars, etc., or into harder metal for rails, as the mill-orders happened to demand, and all from the original ore and the resultant pig, modified only by varying treatment in conversion, etc. I believe the time has arrived when rail-steel must be considered as a special metal, and the blast-furnace must be charged accordingly. Of course, this may cause some inconvenience and slightly increased cost to the rail-makers, but that should not prevent the practice if it is necessary for the production of better rails. The increase of cost would not be very large, and the present condition of the trade does not indicate that the margin of profit is too narrow to permit it. But if such is the fact, then let the rail-makers demand an extra price from the purchasers. If they can present a good case, I have no doubt it would be allowed.

In regard to the fear of an inadequate available supply of low-phosphorus ores, let us consider the figures. It is claimed that nearly 9,500,000 tons of ore will be shipped from the Lake Superior region this season, and fully 60 per cent. of this will be Bessemer ore. Of such ore, 5,700,000 tons would produce at least 3,000,000 tons of steel. The largest amount of rails of all kinds ever produced in this country was 2,139,640 gross tons, in 1887. Since then, 1892 was the heaviest year, with 1,551,844 tons; while 1893 yielded only 1,136,453 tons. In 1892, Illinois made 450,553 tons, and Pennsylvania 961,987 tons. Assuming that the Pennsylvania mills west of the Mountains made 50 per cent. of the State's production, we would have a total western production of 931,447 tons. As there is no immediate prospect of a demand beyond that of 1892, I believe that of the 3,000,000 tons of possible Bessemer steel, not less than one-third can be kept below 0.09 per cent of phosphorus, and thus be available for rails of that composition.

During the past year, quite a large tonnage has been made under practically the conditions of these specifications; and, from the experience had with the rails in the track, and under the drop-tests at the mills, I believe it will be proved that even

a higher percentage of carbon can be used. But it is well not to advance too rapidly. The science of steel-making is steadily progressing. Had the rail-makers, East or West, been asked, a few years ago, for such hard steel as is here called for, it is doubtful whether they would have undertaken the manufacture. Indeed, when the first high-carbon rails were delivered to the N. Y. C. & H. R. R. Co., the maker put himself on record as not being responsible for the damage which would almost certainly occur from their breakage under traffic. Nevertheless, the rails were laid on the sharp curve below Spuyten Duyvil; and I believe that up to this time not one has broken; and they have endured successfully some five years of service.

The only important features in which the present specifications differ from those of 1888, is in providing for a chemical composition and for drop-tests. I have given my reasons for the former, and have added the latter as an additional safeguard, while increasing the hardness of the steel. I still insist on the test-bars being used.

I have left the guaranty as a matter of special understanding between purchaser and maker.

Specifications for Steel-Rails of Heavy Sections Manufactured West of the Alleghanies.

Section.

Sec. 1. The section of the rail rolled shall conform to the template furnished by the railroad company, with an allowance in height of $\frac{1}{64}$ of an inch under, and $\frac{1}{32}$ over, permitted in a delivery of 10,000 tons of rails. The fit of the fishing or "male" template shall be maintained perfect.

Sec. 2. The weight of the rail shall be kept as near to — pounds per yard as is practicable after complying with Sec. 1.

Lengths.

Sec. 3. The standard length of rail shall be thirty feet at a temperature of 60 degrees Fahrenheit. Shorter rails of — lengths will be accepted to the extent of 10 per cent. of the entire order. A variation in length of $\frac{1}{4}$ of an inch longer or shorter than the above specified lengths will be allowed.

Finish.

Sec. 4. The rails must be free from all mechanical defects and flaws, and shall be sawed square at the ends, and the burrs made by the saws carefully chipped and filed off, particularly under the head and on top of the flange.

Sec. 5. The rails shall be smooth on the heads; straight in all directions, both surface and line; and without any twist, waves, or kinks; particular attention being given to having the ends without kinks or drop. The hot-straightening shall be carefully done, so that gagging under the cold-press will be reduced to

the minimum, and so applied that the rails shall not be made "lumpy." None such will be accepted except as No. 2 rails.

Drilling.

SEC. 6. Circular holes —— inch in diameter, shall be drilled through the web at —— inches from the bottom of the flange. The center of the first hole —— inches from the end of the rail; and —— inches from the center of the first to the center of the second hole. These holes must be accurate in drilling in every respect, and left without burrs.

Branding.

SEC. 7. The number of the charge, the name of the maker, the month and year of manufacture, shall be marked in plain letters and figures on the side of the web of the rail, in such a position as not to be covered by the fish-plates when laid in the track.

Chemical Composition.

SEC. 8. The carbon in the 70-pound section shall not be below 0.43 per cent. nor over 0.51 per cent. In the 75-pound section, not less than 0 45 per cent. nor over 0.53 per cent. In the 80-pound section, not less than 0.48 per cent. nor over 0.56 per cent. In the 90-pound section, not less than 0.55 per cent. nor over 0.63 per cent. In the 100-pound section not less than 0.62 per cent. nor over 0.70 per cent.

The phosphorus shall not exceed 0.085 per cent.

The silicon shall not be below 0.10 per cent.

The remainder of the chemical composition of the steel to be left to the makers' judgment.

Tests.

SEC. 9. While the heat is being cast, two test-ingots shall be made; the first from steel going into the first regular ingot; the other from metal representing the last one. These test ingots shall be 3 by 3 inches, and not less than 4 inches long. From these, bars at least ½-inch square shall be drawn at one heat by hammering. Each bar when cold shall be bent, without breaking, to not less than a right angle. Should one bar from a heat fail and the other stand the test, a third bar may be taken from a bloom rolled from the ingot represented by the failed one. If this stands the test, it shall be accepted in lieu of the failed one. If the makers choose, more than the two test-ingots may be taken, but they must be from the steel of the first and last regular ingots. If this is done and a test-bar fail, another one may be drawn from the duplicate ingot and tested, and if it stands, accepted.

Drop-Tests.

SEC. 10. A rail-butt from each conversion shall be placed either head or base upwards on solid steel or iron supports, the distance apart of which, in the clear, shall be 3 feet for sections up to and including 70 pounds, and 4 feet for all heavier ones; and upon it shall be dropped a weight of 2000 pounds falling freely from a height of 16 feet for 70-pound, and 20 feet for all heavier rails. Should a test fail to stand the drop without breaking, a second one may be made. If it also fails, all rails made from that heat shall be rejected; but if the second test stands, then a third one shall be made, and if this be successful, the rails of that conversion shall be accepted.

Treatment of Ingots, etc.

SEC. 11. After the ingots are cast, they shall be either constantly kept in an upright position until ready to be rolled, or else so maintained until the interior steel has had time to solidify.

SEC. 12. No "bled" ingots, or ingots from "chilled" heats, shall be used in the manufacture of rails under this contract.

SEC. 13. No ingots from badly-teemed heats shall be used, excepting as they shall be subject to the provisions of Section 17.

Cutting of Blooms.

SEC. 14. After cutting off, or allowing for the "sand," or top-end of each ingot, at least 12 inches more of seemingly solid steel shall be cut off that end of the bloom—a greater length than 12 inches being preferred; and if, after cutting such length, the steel does not look solid, the cutting shall continue until it does.

Heating.

SEC. 15. Care shall be taken to avoid overheating the steel, and under no circumstances shall a "cinder"-heat be allowed—that is, a heat high enough to cause the cinder to run off the steel as it is being drawn from the furnace. This does not apply to cinder which may be sticking to the under-side of the steel when drawn from a horizontal furnace, or to the bottom of an ingot when drawn from a soaking-pit.

Inspection.

SEC. 16. Inspectors representing the purchaser shall have free entry to the works of the makers at all times while this contract is being filled, and shall have all reasonable facilities afforded to satisfy them that the rails are being made in accordance with these specifications. The makers shall furnish them with the carbon-determinations of each heat, and a sufficient number of complete analyses to represent the average steel of each day's work.

SEC. 17. The inspectors shall have authority to reject rails made from insufficiently-sheared blooms, or from heats the test-pieces or drop-tests of which have failed, or from badly-poured heats, or from "chilled" heats, or from "bled" ingots. The rails made from insufficiently-cut blooms, if otherwise perfect, to be afterwards received as No. 1 short rails, if sufficient lengths have been sawed off to make an amount of steel equal to the original demand of 12 inches. The rails made from heats the test-pieces or drop-tests of which have failed, may be accepted as No. 2 rails. The rails from a badly-poured heat may be received as No. 2 rails, but if made from a "chilled" heat or "bled" ingot, are to be absolutely rejected. By a badly-poured heat is meant one which, from any cause, has been teemed without the control of the operator. A "chilled" heat is one which, by reason of the chilling of the steel, has to be either pricked or poured over the top of the ladle. A "bled" ingot is one from the centre of which the liquid steel has been permitted to escape.

SEC. 18. Imperfectly-drilled, straightened (except "lumpy" rails) or chipped and filed rails shall be rejected, but will be accepted after being properly finished.

SEC. 19. Rails failing to comply with Section 1 will be rejected as No. 1 rails.

No. 2 Rails.

SEC. 20. The requirements for No 2 rails shall be the same as for No. 1, except that they will be accepted with a flaw in the head not exceeding ¼-inch, and flaws in the flanges not exceeding ½-inch in depth, and may have been made from an imperfectly-poured ingot or from heats of which the test-bars or drop-tests have failed.

SEC. 21. No. 2 rails to the extent of —— per cent. of the whole order will be received.

Guaranty.

[Left to special agreement.]

The Present Condition of Gold-Mining in the Southern Appalachian States.

BY H. B. C. NITZE AND H. A. J. WILKENS, BALTIMORE, MD.

(Atlanta Meeting, October, 1895.)

CONTENTS.

I.—INTRODUCTORY.

FROM time to time papers treating of specific cases of Southern gold-mines and mining have appeared in the *Transactions* of this Institute, as well as in other journals and publications. Mr. George F. Becker, of the United States Geological Survey, made a reconnoissance of the Southern Appalachian gold-belt during the fall of 1894; the results of his investigations are published under the title of "A Reconnoissance of the Gold-Fields of the Southern Appalachians," in the *Sixteenth Annual Report of the Director of the United States Geological Survey*, 1894–95: Part III., "Mineral Resources of the United States." Mr. Becker has treated the subject mainly in a scientific geological manner. We present this paper to the members of the Institute as requested by the Secretary last May, with the object of covering the ground from a more practical and technical standpoint. The notes for this paper were made in a large measure from personal observations, gained chiefly during the past summer on a trip through the field. In connection with this trip we collected a suite of typical hand-specimens, representing the gold-ores and wall-rock from the more important localities. This collection forms a part of the exhibit

at the Exposition,* and may serve as a supplement to this paper. The limited time and space at our command does not permit us to enter into the subject in all of its various aspects and details, as its extensive scope might otherwise warrant. It may be in place here to say a few words explanatory of the arrangement of this paper. A bibliography has been omitted, and we would refer the members to an excellent one compiled by Mr. Becker for his paper. The historical chapter is intended simply as a short sketch of the development of the region, and of the mining and metallurgical methods. In the part entitled, "General Distribution of the Mines, with Notes," we have attempted to give a general idea of the extent and location of the principal mining operations in the Southern States; the list of mines given is not a complete one, but represents, so far as can be ascertained, those at present working as well as a large number which during a portion of their career attained some prominence. To facilitate their location, these mines are given by States and counties, and in general order from north to south, so far as possible. Whenever our notes and information were full enough, a concise description of the mines is given. Unless stated otherwise, the mines are not at present working. In the part entitled, "Description of the Mining, Milling and Metallurgical Practice at some of the Specially Characteristic Mines," we have attempted to give a more detailed description of such mines as we judged to represent characteristic classes, from placer- to vein-mining with milling and chlorination. These typical cases do not indicate a general average of the mining operations in the South; they are mines of more or less prominence, and such as afforded better facilities for gaining information. The description of the Haile mine in South Carolina was written in co-operation with Mr. A. Thies, to whom our thanks are also due for much other information on the gold-belt. Captain John Wilkes of the Mecklenburg Iron Works, Charlotte, North Carolina, as well as others, have kindly assisted us by the loan of drawings and maps. We desire, furthermore, to express our sincere acknowledgment to many gentlemen throughout the region for their ready aid and hospitality.

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* *The Cotton States and International Exposition at Atlanta, Ga.*, September 17 to December 31, 1895.

II. Geographical and Geological Description of the Gold-Belts.

The gold-fields of the Southern Appalachians are situated in the area of the crystalline rocks (schistose and massive) extending from the vicinity of Washington in a general southwesterly direction, through the Piedmont and mountain regions of Maryland, Virginia, North Carolina, Tennessee, South Carolina, Georgia and Alabama, to the vicinity of Montgomery. The greatest width of the belt, as a whole, is attained in North Carolina, South Carolina and Georgia, where it is from 100 to 150 miles, narrowing down in Virginia and Maryland on the northeast and Alabama on the southwest (see map, Fig. 1).

In that part of this paper headed "General Distribution of the Mines, with Notes," the gold-mining counties of these States are given.

The general term crystalline rocks includes gneisses, argillaceous, hydro-micaceous, chloritic, siliceous and other schists and slates, limestone, granite, diorite, diabase and other eruptives, as well as certain volcanic porphyries, etc., and pyroclastic breccias. The age of these rocks is Archean, Algonkian, and possibly in part Paleozoic. On the east they are covered by the Coastal Plain and in places by small patches of the Jura Trias (Newark), which latter also occur within the area in small isolated basins, notably in Virginia. On the west they are bordered by the Paleozoic rocks.

The rocks of the gold-belt are decomposed to depths often reaching 50 and 100 feet. Mr. Becker has proposed and used the term "saprolite,"* signifying literally "rotten rock," as a general name for such thoroughly decomposed, earthy, but untransported, rock.

For geological reasons and for descriptive convenience this gold-belt of the Southern Appalachians is differentiated into six minor belts, viz.:

(1) The Virginia Belt.
(2) The Eastern Carolina Belt.
(3) The Carolina Belt.

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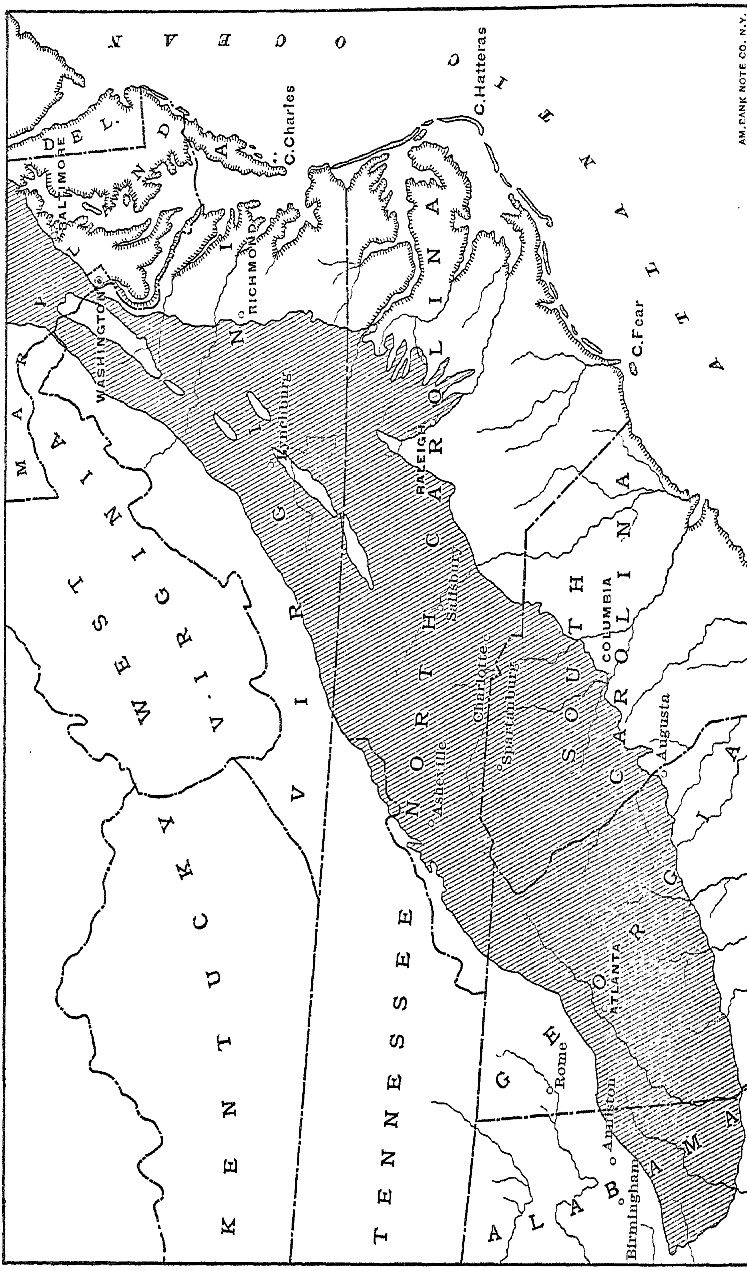
* "Reconnoissance of the Gold-Fields of the Southern Appalachians, by G. F. Becker, *Sixteenth Annual Report of the U. S. Geological Survey*, 1894–5, pp. 289–90.

FIG. 1.



Gold Belt of the Southern States. Scale, 110 miles = 1 inch. Shaded portion shows area of crystalline schists.

(4) The South Mountain Belt.
(5) The Georgia Belt.
(6) The Alabama Belt.

A still further subdivision might be made, as, for instance, the isolated belts west of the Blue Ridge in Virginia, North Carolina and Tennessee and a lower belt in Georgia, in Meriwether and other counties; but not sufficient is known of these at present to say much about them.

The geological description of these various belts can only be briefly taken up here. For fuller and more detailed descriptions the reader is referred to the following papers:

"Reports on the Survey of South Carolina," by O. M. Lieber, Columbia, S. C., 1856, 1857 and 1858.

"A Reconnoissance of the Gold-Fields of the Southern Appalachians," by George F. Becker.*

"The Gold-Ores of North Carolina," by H. B. C. Nitze.†

"The Lower Gold-Belt of Alabama," by William B. Phillips.‡

Work has been in progress by the Geological Surveys of Georgia and Alabama on the gold-fields, and reports from these respective bureaus are expected to be published shortly.

1. *The Virginia Belt.*

This belt begins in Montgomery county, Maryland, and extends in a southwesterly direction, parallel to and on the east side of the Blue Ridge, to the North Carolina line. The best and most reliable, though incomplete, information regarding the geology of this region is given in the early reports of Prof. William B. Rogers (1835, 1836 and 1840).§

The width of the belt is from 9 to 20 miles, and its best developed portion is in Fauquier, Culpepper, Stafford, Orange, Spottsylvania, Louisa, Fluvanna, Goochland and Buckingham counties.

The Country-Rock.—The rocks of the Virginia belt are mica-gneisses and schists, often garnetiferous, talcose (?) and chloritic. The strike is about N. 20°–30° E., and the dip easterly

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* *U. S. Geological Survey, Sixteenth Annual Report,* 1894–95, part iii., pp. 250–332.

† *Geological Survey of North Carolina,* Bull. No. 3, 1895.

‡ *Geological Survey of Alabama,* Bull. No. 3, 1892.

§ *The Geology of the Virginias,* D. Appleton & Co., New York, 1884, pp. 74–80, 131–132, 458–460.

at varying angles. Mr. S. F. Emmons* gives the prevailing strike in Montgomery county, Maryland, as north and south, and the dip nearly vertical or very slightly inclined to the eastward. Granite and diabase dikes occur in the region, and these are sometimes sheared. In some private notes on the Arminius pyrite-mine, in Louisa county, Va., Mr. Becker says:

"The principal country-rock is a series of micaceous schists. . . . . Indications are not wanting that a portion of these schists is of sedimentary origin. . . . . On the other hand, it is equally certain that the most prominent characteristics of the schists are of dynamic origin. . . . . Much of the schist looks as if it were derived dynamically from granite."

The Quartz-Veins.—The auriferous quartz-veins conform in the main to the strike and dip of the enclosing rock. However, their origin is not coeval, the schistose structure antedating the formation of the veins. Neither must their approximate conformity to the country be taken in the absolute sense, for they often cut the schists at small angles both in dip and strike. The structure of the veins is irregularly lenticular, varying from a few inches to several feet in thickness. The wall-rock is often impregnated with auriferous pyrites to considerable extent. Some of these veins are of remarkable persistency and continuity, as, for instance, the Fisher lode in Louisa county, which has been opened for a distance of some five miles along the strike to a maximum depth of 220 feet by the Warren Hill, Louisa, Slate Hill, Luce and Harris mines.

The gravel placer-deposits of the Virginia belt are in all respects similar to those of other gold-regions.

A small isolated gold-belt is situated on the west side of the Blue Ridge in Montgomery, Floyd and Grayson counties, but it is of little economical importance and will not warrant more than this passing mention. The auriferous copper-ores of Ashe and Watauga counties, N. C., also appear to belong here.

2. *The Eastern Carolina Belt.*

This forms a small and narrow area in Halifax, Warren, Nash and Franklin counties. It is covered on the east by the Coastal Plain and bounded on the west by the Louisburg granite. The country-rock is diorite, in great part sheared to a chloritic schist (as at the Mann-Arrington mine). The strike

* "Notes on the Gold-Deposits of Montgomery county, Md.," by S. F. Emmons, *Trans.*, xviii., 391–411.

of the schists is N. 50°–60° E., and the dip 25°–40° S. E. Other intrusives, such as diabase, occur in the region.

The Quartz-Veins.—These veins occur (1) as lenses, from minute size up to 12 inches in thickness, interbedded in the schists or cutting them at low angles; (2) as a reticulated net-work in the massive rocks. It is stated that the saprolites are auriferous over large areas and will repay hydraulic mining.

3. *The Carolina Belt.*

This belt is one of the most extensive and important in the Southern Appalachians, though lying far to the east of the Blue Ridge. It is situated in the central Piedmont region, and extends from the Virginia line in a southwesterly direction across the central part of North Carolina into the northern part of South Carolina, where it sinks beneath the Coastal Plain, making its re-appearance in Abbeville county, S. C., and in Wilkes, McDuffie and adjacent counties in Georgia, near Augusta. There are no mountain chains in the Carolina belt, the only prominences of consequence being a low range of hills known as the Uharie mountains, in Montgomery county, N. C., and the isolated peaks of Crowder's and King's mountains in Gaston county, N. C., extending into York county, S. C.

The belt varies in width from 8 to 50 miles; it is bounded on the west by an extensive granite area,* and on the east by the Jura Trias (Newark) and the Coastal Plain formations.

The Country-Rocks.—The gold-bearing rocks of the Carolina belt are (1) argillaceous, sericitic and chloritic metamorphosed slates and schists; (2) devitrified ancient volcanics (rhyolite, quartz-porphyry, etc.), and pyroclastic breccias; (3) igneous plutonic rocks (granite, diorite, diabase, etc.); (4) siliceous magnesian limestone; (5) sedimentary pre-Jura Trias slates. The Jura-Trias conglomerates along the eastern boundary have also been found to contain gold, but not in quantities of economical importance.

The argillaceous and sericitic† slates, though in general highly metamorphosed and sheared, show many evidences of

* The eastern edge of this granite area contains auriferous veins. The Carolina belt might thus more properly be divided into the Carolina slate and the Carolina granite belts. For various reasons, however, they are included together here.

† The general term "talc" schists, so often used, is very loosely applied, and

sedimentary origin. The siliceous magnesian limestones (King's mountain, etc.), must be included here. They are non-fossiliferous and must be provisionally classed as Algonkian. They are often silicified in varying degrees up to a completeness which renders the rock so hard that it resists scratching with a knife. The chloritic schists are more truly the crystalline schists, and probably represent the sheared basic eruptives. They are even porphyritic and brecciated in places. They are not so abundant as the argillaceous schists, and are richer in accessory metamorphic minerals, such as garnet and epidote.

The general strike of the schistosity is N. 20°–55° E., and the predominating dip to the N.W. from 55°–85°. In many cases, the force producing schistosity and slaty cleavage appears to have acted downward from the N.W., developing normal faulting with but little deformation.

The volcanic rocks occupy irregular patches along the eastern border of the belt, in close proximity to the western edges of the Jura-Trias basins. They comprise both acid and basic types. The acid rocks are generally devitrified to such an extent that their real character is no longer recognizable to the naked eye, and they appear as true cherts or hornstones, although flow-structure is at times still discernible. Microscopic examination shows them to belong to the class of rhyolites and quartz-porphyries. They are sometimes sheared into schists, as for instance at the Haile mine, S. C. The basic types are dark green in color, and perhaps pyroxenic in composition; they are sometimes massive porphyrites, but more generally sheared into schists. The pyroclastic breccias consist of this basic material, in which are imbedded angular fragments of the acid rhyolites and porphyries. The age of these ancient volcanics is believed to be pre-Cambrian. They seem to be analogous to, and probably contemporaneous with, similar rocks of the South mountain in Maryland and Pennsylvania, and other points along the Atlantic coast. The igneous plutonic rocks lie on the western side of the central slates; they consist of granites, diorites, gabbros, diabases, etc. In point of age they are supposed to be younger than the slates and schists on the east. Diabase dikes are common in the Carolina belt, and appear in

generally incorrectly, as the true "talc" schists are comparatively rare; it should more properly be hydro-mica or sericite schists, from a mineralogical standpoint.

general to have exercised a favorable influence on the richness of the ore-bodies which they intersect; the ores often are richer in the vicinity of the dikes. At the Haile mine, in Lancaster county, S. C., this is very marked.

The sedimentary pre-Jura Trias slates, mentioned above as the fifth class of gold-bearing rocks, are perhaps best developed near Monroe, Union county, N. C., and have been called the Monroe slates. These slates are but little indurated, and lie in flat-bedded alternating synclinals and anticlinals. They cover a considerable area, extending from Monroe northward and eastward, and appearing in Stanley and Montgomery counties. They dip under the Jura-Trias conglomerate near Polkton, 20 miles east of Monroe, and might be looked upon as Lower Paleozoic; but the absence of fossils, so far as present search has gone, must, for the time being, place them provisionally in the Algonkian.

The Gold-Ores.—The gold-ores in the Carolina belt exist in two principal structural forms: (1) as quartz fissure-veins; (2) as pyritic impregnations, accompanied by irregular stringer-like and lenticular quartz intercalations in the country schists and slates. The fissure-veins in the slates and schists are generally difficult to distinguish as such. Their structure is much more evident in the granitic and other eruptives. In the schists the larger and more regular quartz-lodes lie apparently interlaminated with the country, or have the appearance of lenticular intercalations; however, even here they can usually be shown to intersect the schistosity, generally at very low angles.

The age of the ore-deposits is later than that of the force which produced schistosity, from the fact that fragmental inclusions of sheared country-rock are not rare in the quartz. The fissuring-force was, therefore, subsequent to the shearing-force. Certain maximum lines of faulting may have been developed, which made room for the larger fissure-veins, on either side of which smaller dislocations formed belts of variable width. It is certainly most natural that, in a rock like slate or schist, the rupturing force should have been exerted along the lines of least resistance, that is, along the cleavage-planes, and that the predominating fissures should, therefore, have been formed in that direction. Isolated instances of cross-fissures occur, but they are rare.

A very usual occurrence of the ores is that of irregular, finely-divided disseminations of auriferous sulphurets and fine gold, accompanied by small stringers and lenses of quartz in the country slates and schists, which are usually silicified, at least to some extent. This form of deposit bears close resemblance to the Scandinavian "fahlbands," which are described as belts of schists impregnated with sulphides. In the Southern Appalachian field they form the small and large bodies of low-grade ores (Haile mine, Russell mine, etc.). The shape of these ore-bodies is lenticular; their outline, however, does not necessarily conform with the strike and dip of the schists, but is determined rather by the degree of impregnation. Very often, also, the wall-rock of the quartz fissure-veins is impregnated for some distance with auriferous sulphurets.

The gravel-placers of the Carolina belt present no features differing from those of similar deposits in other gold-regions.

Genesis of the Ore-Bodies.—No definite proof of metasomatic formation of the ores has been observed; and the most reasonable hypothesis for their formation is that of the ascension and percolation of heated carbonated and alkaline waters carrying silica, metallic elements and sulphides in solution, and the deposition of their mineral contents in the open spaces through which they circulated, by relief of pressure, reduction of temperature, and, perhaps, certain chemical reactions. The frequent silicification of the slates and schists has been noted, and must be ascribed to this permeation of the silicified waters.

The character of the quartz varies from saccharoidal to vitreous, usually inclining to the latter. The sulphurets are chiefly pyrites; chalcopyrite, galena, mispickel and zinc-blende occur in certain localities, notably at the Silver Hill and Silver Valley mines, in Randolph county, N. C. Copper-ores (chalcopyrite) in some of the North Carolina mines are auriferous to such an extent as to make them valuable for gold also, as for instance at the Conrad Hill. Tellurides have been found in very small quantities, as at the King's Mountain mine, N. C. Among the more common gangue-minerals, besides quartz and sulphurets, are chlorite, barite and carbonates.*

* Mr. Becker tabulates no less than 60 gangue-minerals, besides quartz, pyrite, and the ordinary products of decomposition.

The Age of the Ore-Deposits.—The formation of the ores took place subsequent to the production of schistosity. The fact that the Jura-Trias conglomerates, on the east, contain gold, proves that the origin of the gold must have been pre-Jura Triassic. The presence of gold-bearing fissure-veins in the Monroe slates shows that their age must be Algonkian or later. The existence of ore-bodies in the pre-Cambrian volcanic rocks furnishes another clue; and thus it becomes probable that the age of the gold-ores in the Carolina belt is Algonkian.

4. *The South Mountain Belt.*

This belt is situated in the western part of North Carolina, and takes its name from the South mountains, one of the eastern outliers of the Blue Ridge. The principal mining-region embraces an area of 250 to 300 square miles, in Burke, McDowell and Rutherford counties, extending from Morganton to near Rutherfordton, a distance of about 25 miles, with an average width of 10 to 12 miles. The gold-veins of northern Burke and Caldwell counties on the north, and Cleveland and Polk counties, N. C., on the south, as well as Spartanburg, Greenville and Pickens counties, S. C., might be considered as belonging to this general belt; but no extensive operations have been carried on there.

The Country-Rock.—In the South Mountain region, the crystalline schists are for the most part Archean mica (biotite) and hornblende-gneisses and schists, having an eminently lenticular structure. They are often garnetiferous and contain also many rare minerals, such as zircon, monazite, xenotime, etc. These gneisses are considered to have been igneous granites and diorites, subsequently rendered schistose by dynamo-metamorphism. The general strike of the schistosity is N. 10°–25° W., and the dip 20°–25° N.E. To the northwest of South Muddy creek and Vein mountain, however, the strike is generally N.E. and the dip S.E. This is the case also in the northern part of the general belt, in Caldwell county.

Isolated masses of pyroxenite and amphibolite occur as rounded inclusions or blebs, from less than 1 to nearly 100 feet in diameter, in the gneiss. They are looked upon as basic segregations from the original igneous magma out of which the gneisses were formed. They alter to talc and serpentine.

Pegmatites are of frequent occurrence in the gneisses, and like them their structure is usually lenticular. At several points there are indications of pegmatite dikes. Granite dikes occur in the South Mountain region; and in the northern part of the belt, in Caldwell county, a very persistent and continuous dike of aphanitic olivine diabase has been observed. Brown mountain, in the northern part of Burke county, is made of granite.

The Quartz-Veins.—The auriferous quartz-veins of the South Mountain belt form a system of parallel fissures of remarkable regularity, striking N. 60°–70° E. and dipping 70°–80° N. W. Their thickness varies from that of a knife-edge to 4 feet. The great majority are from less than 1 to 3 inches in thickness, lying in zones of scores of small veins; the larger ones (1 to 4 feet) are few and far between. Normal faulting has been observed in a few instances. The ore is quartz, usually of a milky white color, generally saccharoidal and seldom vitreous or glassy. It is often stained brown and is cellular from decomposed sulphurets. The sulphurets are pyrite, galena, chalcopyrite, and zinc-blende. All observations go to show that the vein-matter is formed from ascending mineralized solutions. There is no evidence of the replacement of the country-rock by ore.

In the South Mountain region proper there are five parallel lines or zones, along which these quartz-veins appear to be concentrated in force:

1. The Morganton zone, passing through Morganton, along Little Silver creek and through the Neighbor's Place to North Muddy creek.
2. The Huntsville zone, passing over the southern end of Huntsville mountain.
3. The Pilot Mountain zone, passing over Hall's knob, White's knob, Pilot mountain, Brackettown and Vein mountain, to and beyond the Second Broad river.
4. The Golden Valley zone, passing across the upper end of the Golden valley (valley of the First Broad river) and crossing Cane and Camp creeks to the Second Broad river.
5. The Idler Mine zone, about 3 miles north of Rutherfordton.

The great majority of these auriferous quartz-veins are too

small to be profitably worked individually. Of the larger and more promising veins which have been worked, the "Nichols," at Vein mountain (18 inches to 3 feet), and the "Idler," near Rutherfordton (22 inches), may be mentioned.

The Placer-Deposits.—The principal mining ground of the South Mountain region is that of the placer-deposits. These are of three classes: 1. The gravel-deposits of the stream-beds and bottom-lands, deposited by fluviatile action. 2. The gulch and hill-side deposits, or accumulations due to secular disintegration and motion induced by frost-action and gravity. 3. The upper decomposed layer of the country in place, the saprolites. In the first class the gravel is water-worn, rounded to subangular, and the deposits are from 1 to 2 feet in thickness. In the second class the gravel is usually quite angular, and the deposits are from a few inches to several feet in thickness. In the third class, gravel is of course absent, the washable ground consisting of the upper decomposed layer in place, the gold being derived directly from the partially disintegrated quartz-veins.

Minor Belts.—On the west side of the Blue Ridge, in Henderson county, N. C., gold has been mined at the Boylston mine. The country-rocks are fine-grained mica- and hornblende-schists, in part much crumpled. The general strike is N. 20°–30° E., and the dip N.W. The schists are cut by a granite dike. The valley of Boylston creek is made up of schistose limestone, underlying these crumpled schists. These rocks are probably to be classed in the Ocoee, which by some is supposed to be Algonkian and by others Paleozoic. This isolated belt, however, has little economic importance.

Another belt of auriferous rocks is that in which some unimportant placer-mining operations have been prosecuted in Swain, Jackson and Cherokee counties, N. C. The country-rock is supposed to be largely Ocoee. In Tennessee the petty stream deposits of Polk, McMinn, Monroe and Blount counties are probably in the same formation.

5. *The Georgia Belt.*

The Georgia belt is probably of next or equal economic importance to the Carolina belt. Beginning in Rabun and Habersham counties, in the northeastern corner of the State, it extends in a southwesterly direction through the important

mining town of Dahlonega, and thence to the Alabama line in the vicinity of Tallapoosa. This is in the Piedmont region of the State, lying on the southeast side of the Blue Ridge. Although the maximum width (N.W. and S.E.) over which the mines are distributed is as great as 30 miles, the principal portion of the belt, which extends from near Canton, in Cherokee county, through Dahlonega and Nacoochee, to Clayton, in Rabun county, is concentrated in a width of 4 miles or less. It is to this latter portion that the following geological descriptions more especially relate.

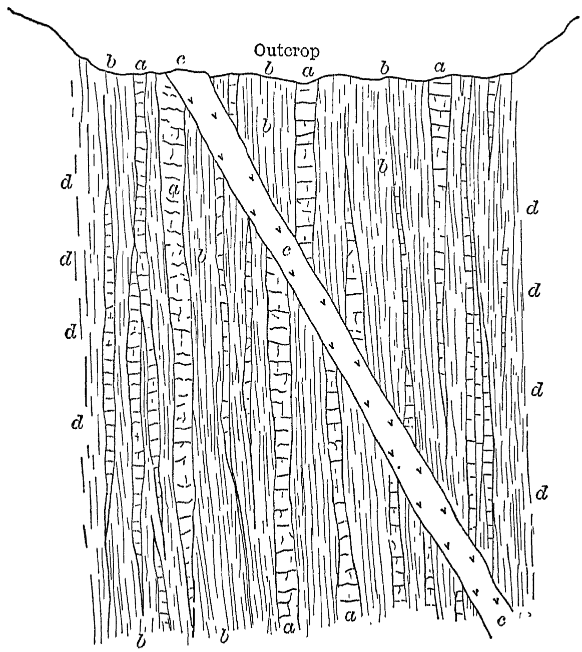
The Country-Rock.—The rocks of this belt resemble in many respects those already described under the South Mountain belt in North Carolina. They are Archean mica- and hornblende-gneisses and schists, which probably represent sheared granitic and dioritic rocks. At the Murray mill, on Yahoolah creek, near Dahlonega, a large mass of unsheared granite may be seen; and massive granite is reported to exist on Yonah Peak, near Nacoochee. These gneisses and schists are banded in narrow, lenticular-shaped layers, from 2 to 20 feet wide. A dark-colored, schistose hornblende-rock, locally known as "brick-bat," is of frequent occurrence. Its structural relations are very difficult to determine; at times it is conformably interlaminated with the other schists (as at the Hedwig mine, near Auraria); again, it appears to have no regular relation in its position to the adjoining schists, which are cut off by it or very markedly disturbed in their strike, bending around the "brick-bat" mass, and developing a crumpled or folded structure in the schistose laminæ (as at the Singleton and Lockhart mines, near Dahlonega). It is possible that these "brick-bat" masses, which appear to be dioritic in origin, are magmatic segregations or blebs, similar to the pyroxenic and hornblendic blebs described in the South Mountain region, though, as a rule, larger. The prevailing strike of the gneisses and schists is N. 20°–30° E., and the dip 30°–60° S.E. Locally, however, in the presence of the dioritic masses, as explained above, this changes to northwest strikes with northeast dips. The rocks are often garnetiferous and contain other rarer accessory minerals, such as monazite,* though to a much less degree than

* At The Glades Post-Office, in Hall county, 10 miles northeast of Gainesville, monazite has been found in some quantity.

in the South Mountain rocks. The depth of the saprolites in the Georgia belt reaches a maximum of about 100 feet.

Diabase dikes, such as are common in the Carolina belt, are not found in the Georgia belt. Granitic dikes are, however, not uncommon in the Nacoochee region.

FIG. 2.



Cross-section in Opening at Thompson Mine, near Nacoochee, Ga. Scale, 1 inch = 2 feet. *a*, quartz ; *b*, slate ; *c*, granite dike ; *d*, wall-rock.

The accompanying sketch (Fig. 2) represents a small pegmatite dike at the Thompson mine, 4 miles west of Nacoochee, showing the development of normal faulting. Similar granitic dikes have been found in Cherokee county, near the Franklin mine. In the Dahlonega district, although no unquestionable well-marked dikes are seen in place, Mr. Becker * calls attention to the possibility that some of the unusually sharply-marked sheets in the gneiss might be intrusive.

The Ore-Deposits.—Certain bands of the gneisses and schists have been fissured and filled with gold-bearing quartz and sul-

* "Reconnoissance of the Gold-Fields of the Southern Appalachians," *Sixteenth Annual Report of the U. S. Geological Survey*, 1894–5, Part III., p. 296.

phurets. These fissures are in the main parallel to the schistosity of the rock, though not uncommonly they cut the same at low angles. To a large extent they are aggregated in a zone of numerous narrow and discontinuous lenses and stringers through more or less definite bands of the gneiss, which, taken altogether, form the vein. This is well illustrated in Fig. 2. Mr. Becker has designated such a system, a "stringer-lead."* In these narrow, sharply-banded gneisses and schists of different material, such as they are in this part of the Georgia belt, it is natural that the fracturing force, once exerted in a certain band, should have been more or less confined to this one, both longitudinally and transversely, the walls of the band forming the walls of the ore-body. This is in fact the case. At times the fissuring is confined to the light-colored mica gneisses, at other times to the dark colored ferromagnesian gneisses and schists. The "brick-bat" schists rarely contain ore-bodies. The thickness of the veins is from less than 3 to as much as 20 feet; they are frequently close together, separated by non-auriferous bands of gneiss; and the total width of the ore-bearing ground reaches as much as 200 feet (Singleton mine, Dahlonega). The extent of fissuring must depend largely on the degree of homogeneity of the material, as well as on the intensity of the fracturing force. Where the rock is of homogeneous composition and the force uniformly exerted, the effect would be a more or less evenly distributed shattering, with few gaping fissures, and the whole mass would be permeated by the gold-bearing solutions, with the formation of auriferous and pyritic impregnations, with some small quartz stringers. At the Hedwig mine, near Auraria, for instance, regular quartz-masses of any size are altogether absent, the ore-body being composed of soft, sandy, mica-gneisses and schists containing only few, small and isolated quartz-stringers. Again, under different conditions, the effect was the production of a large number of small open fissures, inducing the consequent formation of numerous small lenticular quartz-stringers; and such is the usual case in the Dahlonega ore-bodies (see Fig. 2). Or, where the rock mass was of still greater heterogeneity, and the forces of greater or more varied intensity, lenticular fissures

* "Reconnoissance of the Gold-Fields of the Southern Appalachians," *Sixteenth Annual Report of the U. S. Geological Survey*, 1894–5, Part III., p. 283.

have been opened, of such size and extent as to allow a more or less complete filling by solid auriferous and pyritic quartz, from 3 to 14 feet in thickness; while, further along the strike, though the fracturing extends to the same width and the walls hold out, the intervening space of country has simply been shattered, or opened only in small spaces, but was nevertheless filled with pyritic impregnations and quartz-stringers (as at the Franklin mine in Cherokee county, where these barren portions of the vein are called horses). But the leads are continuous, usually for considerable distances. At the Lockhart mine, near Dahlonega, for instance, the Blackmer vein, 3 to 6 feet in thickness, has been opened by a drift 400 feet long. At the Franklin mine, in Cherokee county, the ore-body has been explored by underground workings for 1000 feet, and the continuity of the vein has been traced for three-quarters of a mile by isolated shafts. The regularity of the vein-structure at the Franklin is exhibited by well-defined walls, and by the presence of a soft "gouge" on both the foot and hanging, even where there is no marked quartz-filling.

Small, clean-cut cross-fissures occur in the Georgia belt, as at the Franklin mine, where the filling is chiefly calcite.

The pitch of the ore-bodies in the Georgia belt is as a rule to the northeast. The filling of the fissures is quartz, carrying pyrite, and rarely chalcopyrite. Among the most interesting gangue-minerals may be mentioned garnets, which in cases have been found to be auriferous.* Another occasional, though rare, gangue-mineral is tourmaline. The character of the quartz varies greatly, from very saccharoidal to extremely vitreous types, and from clear transparent to milky-white in color, sometimes smoky.

The genesis of the ore-deposits is best explained by the ascension theory; there is no evidence of substitution. The formation of the ore-deposits was subsequent to the force that sheared the country, from the fact that fragments of the schistose country occur in the quartz.

The character of the gravel placer-deposits in the Georgia belt is similar to that in the South Mountain belt.

There appears to be an isolated gold-belt of very limited ex-

* "Reconnoissance of the Gold-Fields of the Southern Appalachians," *Sixteenth Annual Report of the U. S. Geological Survey*, 1894–5, Part III., pp. 279, 297.

tent in Meriwether county, Georgia. The country-rock is stated to be gneiss, and the quartz to exist in a series of narrow lenses.

6. *The Alabama Belt.*

The Alabama belt might be considered a continuation of the Georgia belt. However, principally as a matter of convenience for reference, it is spoken of and described separately here. It comprises an area of about 3500 square miles, situated in the crystalline rocks of Cleburne, Randolph, Talladega, Clay, Tallapoosa, Chambers, Coosa, Elmore and Chilton counties. This is the southwest extremity of the southern Appalachian gold-field.

On the latest geological map of Alabama,* the gold-bearing rocks of this area are distinguished as: 1. The semi-crystalline Talladega shales of Algonkian age, including argillaceous and hard, greenish, sandy shales (often graphitic); 2. The crystalline schists of Archean age, including mica-schists, which, on the one hand, grade through gneisses into granite, and, on the other, into siliceous schists; garnetiferous hornblende-schists, probably of dioritic origin, also occur. The general strike is N.E. and the dip S.E.

The quartz veins are interlaminated in these rocks, coinciding imperfectly with the dip and strike of the schistosity. From a structural geological standpoint, the veins bear much similarity to those of the Dahlonega type. From a mining standpoint, however, they are different, not forming the wide belts of numerous parallel leads, as in Dahlonega. The quartz is usually glassy; the sulphurets are in the main pyritic, and the gangue-minerals are those of usual occurrence in gold-bearing quartz-veins elsewhere. The character of the placer-deposits presents no novel features.

III.—Historical.

For the probable earliest discoveries of gold in the southern part of the United States by the Spanish explorers we refer the reader to Mr. Becker's treatise. Reports of the existence of gold in the Southern States antedate the time of the Revo-

* Geological Map of Alabama, with Explanatory Chart, *Geological Survey of Alabama*, 1894.

lutionary War, but no authentic references to these can be obtained. Jefferson, in his "Notes on Virginia" (1782), mentions the discovery of a nugget, containing 17 dwts. of gold, 4 miles below the falls of the Rappahannock river. The U. S. Mint Reports give the first returns from Virginia in 1829. For North Carolina, the first mint-returns appear in 1793; but the first mention of any specific find of gold in North Carolina is of a 17-pound nugget discovered on the Reed plantation, in Cabarrus county, in 1799. From 1804 to 1828, North Carolina furnished all the gold produced in this country, amounting to $110,000. Mills, in his "Statistics of South Carolina," notes the occurrence of gold in the Abbeville and Spartanburg districts as early as 1826, but the first U. S. mint-returns from this State are given in 1829. In Georgia, the first discovery is stated to have been in Habersham county in 1829. The earliest mint-returns from this State appear in 1830. Dr. W. B. Phillips* gives 1830 as the probable approximate date of the first discovery of gold in Alabama. There were, however, no mint-returns from this State until 1840. The first mention of gold in Tennessee is from Coker creek, Monroe county, in 1831;† and this date corresponds with that of the first mint-receipts. The earliest record of gold in Maryland is in 1849,‡ from the farm of Mr. Samuel Ellicott in Montgomery county; the mint-reports, however, show no returns previous to 1868.

The greatest activity of gold-mining in the South seems to have followed closely on the first discovery, being most marked from 1829 to 1836, and probably due to the working of the more accessible virgin placers and more easily mined outcrops. The mint-receipts show a renewed activity from 1839 to 1849, caused perhaps by more systematic vein explorations and improved methods. In the early fifties, the Californian discoveries abated the interest in the Southern gold-field and attracted the mining population westward, causing a natural depression in the output; from that time on there was a general decrease until the practically total cessation caused by the Civil War. Since then there have been spasmodic revivals and depressions, due un-

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* *Geological Survey of Alabama*, Bulletin No. 3, 1892, p. 10.

† Safford's *Geology of Tennessee*, 1869, p. 489.

‡ *Proceedings of the American Philosophical Society*, vol. v., p. 84, 1849.

doubtedly in a great measure to local causes and excitements, and to the financial condition of the country at large. Considering the small total output of the South, such fluctuations may have been caused by the successful working of a single mine, shown for instance by the increased production of South Carolina since 1890, owing to the revival of the Haile mine.

It may be of interest here to say something of the mining and metallurgical methods from the earlier times to the present day. The first primitive washing, as in other newly-discovered gold countries, was probably done with the pan. As the workings grew more extensive, this was superseded by the rocker, long-tom, and sluice-box; and these original devices survive to the present day in many localities, operated by tributers and petty miners. The rockers in use to-day are of two types. The first is essentially a panning process, using a minimum amount of water, the operation being an intermittent one. This type of rocker is closed at both ends, the discharge being over the side; it is described below (p. 732), with illustrations, as in use at the Crawford mine. The second type consists of a hollow segment of a log closed at the upper end. It is set on a slight inclination, about 6 inches in 10 feet, and is provided at the lower end with grooves or strips that act as mercury-pockets or riffles. When used on gravel, it is provided at the upper end with a shallow box having a round punched or slotted iron bottom. The length of this type of rocker is about 5 feet. The gravel and clay are thrown into the box, where a constant stream of water, together with the rocking motion and stirring with fork or shovel, disintegrates the material. The pebbles and bowlders are thrown out with the fork, while the fine portions are washed down the bottom. The rocking facilitates the settling and amalgamation of the gold, and the discharge of the tailings. Two men work at one rocker. One throws the gravel from the pit into the box, and the other sits above the rocker moving it with his feet, disintegrating the gravel with a fork, and discharging the coarse material. Rockers of a similar type are at present in use at several mills for handling pulp and blanket-washings (see Gold Hill, p. 706).

Where sufficient flowing water is at hand, the sluice-box and long-tom are used, as they handle larger quantities with

less labor. The sluice-box, generally 8 to 10 feet long, 20 inches wide and 12 inches deep, provided with riffles and a perforated charging-plate at the head, fulfils the same purpose as the rocker; being stationary, however, it requires a larger amount of water to carry off the tailings.

In the early days of cheap and slave-labor, the main gold-output was obtained by the above devices. Farming and gold-digging went, in many cases, hand in hand. When the crops were laid by, the slaves and farm-hands were turned into the creek-bottoms, thus utilizing their time during the dull season. Where mining proved more profitable than planting, the former superseded the latter entirely. In speaking of the Tinder Flats placer, in Louisa county, Va., Silliman says: "Jenkins is in the habit of substituting a fall working in the gold, for which he obtains $1000 annually, as a compensation for his tobacco crop, which he relinquishes in favor of the gold." *

Some of the more prominent localities developed into regular mining-camps, where continuous and extensive operations were carried on. Such districts were, for instance, Arbacoochee and Goldville, Ala.; Auraria and Dahlonega, Ga.; Gold Hill and Brindletown, N. C. In the latter place, it is stated that, just before the California excitement, as many as 3000 hands might have been seen at work on one of the streams of the region.†

The regular mining carried on at this time (after the stream-deposits were exhausted) consisted of sinking pits in the bottoms and raising the gravel by hand-labor. These pits were drained by large vertical bucket-wheels, for which the power was derived from the stream directly, or by flume-lines with over-shot or under-shot wheels. Tuomey,‡ in 1854, mentions ground-sluicing of side-hill deposits, at Arbacoochee, Ala., by aid of a ditch and a series of trenches into which quicksilver was poured. It is probable that this method of working existed even prior to that day. The first mention of hydraulicking is by Lieber,§ who describes it as "a novel and

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* *Report to the President and Directors of the Walton Mining Company.* By Prof. B. Silliman, Jr., Fredericksburg, Va., 1836.

† *Ores of North Carolina,* p. 312.

‡ *Second Biennial Report on the Geology of Alabama,* p. 70, Montgomery, 1858.

§ *Supplementary Report to the Survey of South Carolina,* 1859, p. 154.

singular method," practised previous to 1859, at Pilot mountain in Burke county, N. C. Hydraulic gravel-elevators and dredge-boats were not used until a comparatively recent date. The first account of vein-mining is in 1825, at the Barringer mine, in Montgomery county, N. C. In other localities it probably followed the exhaustion of the richer gravel-deposits; in Virginia, the first vein-mining mentioned is at the Tellurium mine in 1832. For a long time the output was confined to the free-milling brown ores near the surface. The ore was raised by horse-whim and hand-windlass, or even by baskets carried upon the backs of the miners. The most primitive method of milling the quartz was undoubtedly by hand-mortars and panning. This is still carried on by the native tributers in certain districts. It was followed by the introduction of drag-mills (arrastras), Chilean mills, and wooden stamps. As an illustration of some of these earlier milling-methods, the following is taken from a report of the Supervising Committee of the United States Mining Company, in 1835, on their mine near the Rappahannock river, Va.:

"The plant consists of a crushing (rolls) and a vertical mill (stamp-mill) in a building 26 × 36 feet. Both mills are located on the ground floor and are propelled by a water-wheel 11 feet in diameter, with a 11-foot face. The crushing-mill has 3 sets of cylinders 2 feet in length and 15 inches in diameter, the first or upper set fluted, the other smooth. The ore is thrown into a hopper on the upper floor, from which it is conducted, over an inclined shaking-table, to the fluted cylinders, by which it is crushed to a size from ¼ to 1 inch in diameter. The crushed material is equally divided and goes to the two sets of smooth cylinders. By them it is further greatly reduced, ranging from impalpable powder to grains as large as coarse hominy. From these cylinders it falls into a sifter having the fineness and motion of the common meal-sifter, from whence the material which passes through is conducted to 12 amalgamators, constructed upon the principle of the Tyrolese bowls, making from 90 to 100 revolutions per minute. They perform the office of washing and amalgamating. The sand discarded by them, after being washed, is conducted through troughs to the vertical mill, where, being reduced to an impalpable powder, it passes in the shape of turbid or muddy water to another set of amalgamators similar to those above mentioned, and thence to the river. The portion of the ore reduced by the cylinders which passes over the sifters is conducted to the vertical mill, and is treated in the same manner."

The process at another Virginia mine, the Vaucluse, is described in 1847 as follows:*

* *Plan and Description of the Vaucluse Mine, Orange County, Va.* Philadelphia, 1847.

"The machinery consists of a condensing Cornish mining-engine of 120 horse-power; the mill-house contains 6 large Chilean mills; the cast-iron bed-plate of each is 5 feet 6 inches in diameter, and on it are two cast-iron runners of the same diameter, the total weight of the mill being 6200 pounds. The ores, on arriving at the surface, are divided into two classes: 1. The coarse and hard ore for the stamps; 2. Slate and fine ore for the Chilean mills. This is done by means of a large screen. The very large pieces are first broken by a hammer before they are fed to the stamps. All of the ores are ground with water, each mill being supplied with hot and cold water at pleasure. Twelve inches from the top of the bed-plate there is a wide, open mouth, from which the turbid water escapes to tanks. On the south side of the steam-engine is the stamp-house and amalgamation-mill, containing 6 batteries of 3 stamps each; these stamps, with the iron head of 125 pounds, weigh 350 to 380 pounds each. Each battery is supplied with water, and at each blow of the stamp a portion of the fine ore passes out of the boxes through the grates to the amalgamation-room. Here are stationed 18 small amalgamation-bowls of cast-iron, 30 inches in diameter. The bowls are supplied with runners which move horizontally; in the center of these runners is an eye or opening like that in the runner of a corn-mill. The ground or finely-stamped ore, gold, and water pass into this eye, and by the rotary motion of the same are brought into contact with the quicksilver deposited in the centre, forming amalgam. From the amalgamators the pulp passes through 3 dolly-tubs or catch-alls, acting as mercury and gold tubs. After this the whole mass passes to the strakes or inclined planes, where the sulphurets are deposited and the earthy matter washed away. These sulphurets were formerly treated in two heavy Mexican drags or arrastras; but not answering so good a purpose, they have been altered into three heavy Chilean mills."

The collection of amalgam, retorting and melting was practically the same as to-day. The total plant at this mine was valued at $70,000.

Wooden stamp-mills, with iron shoes and die-plate, were used in Georgia as early as 1845. In construction they were similar to the present California mill, with the exception that the stems did not revolve, the cams working in slots or recesses cut into the square stems. These mills may be seen in operation at present in the Nacoochee valley, Ga. They are cheaply constructed, a 10-stamp mill, with water-wheel and building complete, costing about $150. The amalgamation is done on a copper plate of the width of the battery and about 1 foot long. These mills seem to serve the purposes of the tributers and petty quartz-miners, and it is stated that they are operated at a fair profit. Milling on a more extensive scale is carried on in the South by modern stamp-milling machinery, and there are a number of well-constructed plants. Besides mills of Western manufacture, there are two types which are common to the South. One of these is a 750-pound mill built

by the Mecklenburg Iron Works, of Charlotte, N. C., a slight variation of the Western type (see Haile mine p. 778.) The other is the 450-pound Hall mill, which is peculiarly adapted to the ores of the Dahlonega district. The Crawford and Howland mills have been used to some extent, and several Huntington mills are now in operation in the South. At the Haile mine a plant on the Blake system of fine crushing, combined with subsequent wet grinding in Chilean mills, was erected in 1884.* This was soon abandoned and the present stamp-mill was substituted. The first method of concentration noted is that described on p. 683. (Vaucluse mine, 1847.) This was probably followed by buddles, primitive bumping-tables, and more especially by blankets. At the present day the Frue, Embrey and Triumph concentrators are in general use. Of these, the Embrey machine is considered by some to give better results, especially where skilled labor cannot be obtained, and where the sulphurets are not sized. Still, each one of the three finds its strong advocates, and the difference in perfection of concentration obtained by them is probably not material. In some cases—as, for instance, in the Gold Hill district—the finely-divided condition of the gold has led to the re-employment of blankets. At the Reimer mine, North Carolina, a plant was in operation several years ago in which the ore was comminuted in a series of crushers and 26-inch rolls; the pulp was sized into six grades, from 10- to 60-mesh, and each grade treated separately by a Bradford jig. This process is said to have given good results, but the plant was destroyed by fire shortly after its erection, and never has been rebuilt.

As soon as the water-level had been reached in the mines, and the brown ore was practically exhausted, attempts were made to treat the undecomposed sulphurets.

The earliest treatment of the concentrated sulphurets was by regrinding them in Mexican arrastras and Chilean mills, with subsequent amalgamation, as described above for the Vaucluse mine. Lieber† states that a process for roasting sulphurets, with subsequent amalgamation, was introduced by Mr. C. Ringel at a mine near Rutherfordton, N. C., and afterwards practised

* *Trans.*, xvi., 755.

† *Report on the Survey of South Carolina for* 1856, p. 47.

with entire success on old tailings at the Gold Hill mine and others in North Carolina. It is impossible to follow out the gradual development of the treatment of sulphurets. A large number of processes were tried with more or (generally) less success. Among these may be mentioned specially the Designolle process, worked for a time at a custom-plant near Charlotte, N. C., at the New Discovery mine, Rowan county, N. C., and at the Haile mine,* S. C., but soon abandoned for lack of success. The South has been the "proving-ground" of almost all the patent gold-saving processes invented. It would be impossible and useless to give a list of these. It was not until about 1879 that the successful treatment of sulphurets was accomplished.

This was effected by the Mears chlorination-process, first introduced at the Phœnix mine, Cabarrus county, N. C., under the management of Mr. A. Thies. It was later developed by him into what is now universally known as the Thies process (see Haile mine p. 781.) Some experiments have lately been made by Mr. G. P. Lidner at the Brewer mine., S. C., and in Dahlonega, Ga., with a chlorination-process for treating the ore in bulk. It is stated that this process proved successful at the Findley mine in Georgia, but that the results obtained at other mines were not favorable. A plant for a patent electrolytic chlorination-process has lately been erected at the Clopton mine, Villa Rica, Ga. In the prospectus of this company, entitled "Gold, How to Get it and How to Save it," it is asserted that by the use of this process the South will once more assume her former station in the mining world and become the scene of active and profitable mining operations, rivalling all she has done in the past!

The cyanide-process has so far found but little application in the South. In May, 1892, Mr. Richard Eames, of Salisbury, N. C., experimented with cyanide at the Gold Hill mine, N. C., extracting 60 per cent. of the assay-value. In the summer of 1893, a 10-ton cyanide-plant was working at the Moratock mines, Montgomery, N. C., but the operations were soon relinquished here on account of the low grade and character of the ore. Later in the same year, a cyanide-plant was in operation at the Gilmer mines in Goochland county, Va., with what

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* E. G. Spilsbury, *Trans.*, xv., 771.

success could not be ascertained. At the Franklin mine, Ga., a treatment of the ores with cyanide was attempted before the introduction of the chlorination-process. It proved successful on the oxidized tailings from the old dumps; but the extraction from fresh sulphurets was insufficient to warrant its continuation. The American Cyanide Gold and Silver Recovery Company, of Denver, Colo., has lately experimented on the ores from the Russell mine, Randolph county, N. C. These experiments are stated to have been successful, and a cyanide-plant is now in process of erection at this mine.

A plant for the extraction of gold from sulphurets, with the recovery of the sulphuric acid, has recently been erected at Blacksburg, S. C., mainly for the treatment of custom-ores. The process consists in roasting the concentrated ores in a Walker and Carter muffle-furnace, which is connected with lead chambers. The amalgamation of the roasted concentrates is carried on by a patent process known as the Caloric Reduction Company's process. It is proposed to use the tailings for the manufacture of red paint. Regarding smelting-processes, probably most has been done in the attempted treatment of the complex ores from the Silver Valley and Silver Hill mines, N. C. During the past ten years a number of patent processes have been experimentally tried at Thomasville, N. C., but it was not until about two years ago that a successful process was introduced there by Mr. R. F. Nininger, of Newark, N. J. (see Silver Valley mine, p. 698). Experiments on matting auriferous sulphides from the Haile mine were made in 1886* but proved unsuccessful. Matte-smelting, and the Hunt and Douglas leaching-process, were practised on the copper-ores of Conrad Hill, N. C.

IV.—Statistical.

Table I. is given below to show an estimate of the gold and silver production of the Southern States down to the present time. It is taken from the production-report of the Director of the Mint for 1892, with additional estimates from the same source for the succeeding years. The figures represent not only the amounts deposited at the United States Mint and Assay Offices, but also an estimate of the gold and silver that was produced but not turned in to the mint.

* *Trans.*, xv., 767–775.

TABLE I.—*Estimate of the Production of Gold and Silver in the Southern States from 1799 to 1879 and Annually Since.*

Years.	Md.	Va.	N. C.	S. C.	Ga.	Ala.	Tenn.	Total.
1799-1879	$2,500	$3,091,700	$19,659,600	$2,587,900	$14,180,500	$365,300	$155,300	$40,042,800
1880......	250	11,500	95,000	15,000	120,000	1,000	1,500	244,250
1881......	500	10,000	115,000	40,000	125,000	1,000	1,750	293,250
1882......	1,000	15,000	215,000	25,000	250,000	3,500	250	509,750
1883......	500	7,000	170,000	57,000	200,000	6,000	750	441,250
1884......	500	2,500	160,500	57,500	137,000	5,000	300	363,300
1885......	2,000	3,500	155,000	43,000	136,000	6,000	300	345,800
1886......	1,000	4,000	178,000	38,000	153,500	4,000	500	379,000
1887......	500	14,600	230,000	50,500	110,500	2,500	500	409,100
1888......	3,500	7,500	139,500	39,200	104,500	5,600	1,100	300,900
1889......	3,500	4,113	150,174	47,085	108,069	2,639	750	316,330
1890......	16,962	6,496	126,397	100,294	101,318	2,170	1,001	354,638
1891......	11,264	6,699	101,477	130,149	80,622	2,245	519	332,975
1892......	1,000	5,002	90,196	123,881	95,251	2,419	1,006	318,755
1893......	114	6,190	70,505	127,991	100,375	6,362	250	311,787
1894......	978	7,643	52,927	98,763	99,095	4,092	329	263,827
Total..	$46,068	$3,203,443	$21,709,276	$3,581,263	$16,101,730	$419,827	$166,105	$45,227,712

TABLE II.—*Statement of Gold and Silver produced in the Southern States; Deposited at the United States Mint and Assay Offices from 1793 to 1879 Inclusive.*

Year.	Amount.	Year.	Amount.	Year.	Amount.
1793-1823	$47,000	1842	$723,761	1861	$141,778
1824	5,000	1843	1,050,100	1862	6,298
1825	17,000	1844	928,095	1863	1,624
1826	20,000	1845	986,849	1864	6,093
1827	21,000	1846	992,792	1865	33,345
1828	46,000	1847	1,018,079	1866	202,000
1829	140,000	1848	850,692	1867	106,903
1830	466,000	1849	891,968	1868	155,660
1831	519,000	1850	658,605	1869	191,738
1832	678,000	1851	500,539	1870	168,057
1833	868,000	1852	711,449	1871	138,791
1834	898,000	1853	486,184	1872	164,461
1835	686,300	1854	323,489	1873	158,952
1836	667,000	1855	362,349	1874	141,647
1837	282,000	1856	325,820	1875	150,612
1838*	358,750*	1857	141,810	1876	138,256
1839	429,648	1858	349,323	1877	159,009
1840	427,311	1859	379,677	1878	162,925
1841	544,661	1860	231,398	1879	186,123

In order to give an idea of the fluctuation during the eighty years from 1799 to 1879, Table II. is given. These figures, however, comprise only the United States Mint and Assay Office receipts, and do not include such bullion as went abroad,

* The years 1838 to 1847 exclude the amounts deposited at the New Orleans Mint, which were not available for each year. The total amount at New Orleans in those years from the Southern States was only $116,086.

was sold directly to local jewellers, or was coined by Christian Bechtler* at Rutherfordton, N. C.

The following estimates regarding the Southern gold-mining industry for the year 1889 are taken from the United States Census Report of 1890:†

Total number of producing mines (of these only 8 were producing between 10,000 and 50,000 dollars per year),	42
Total number of tons of ore produced,	123,745
Total yield of gold bullion,	$318,261
Grand total of expenses, wages, etc.,	$535,285
Total value of mines and plant,	$5,281,801
Total value of mills and reduction-works,	$620,681

V.—General Distribution of the Mines, With Notes.‡

Maryland.

The gold-mines of Maryland are situated within the belt of crystalline rocks extending from Washington to Great Falls, on the Potomac river, in Montgomery county. Geologically, they are included in the Virginia belt (see p. 665).

The greatest development has been in the vicinity of Great Falls, about 15 miles west of Washington. Among the principal mines in this region are the Maryland, Montgomery, Harrison, Irma, Huddleston, Allerton-Ream and Sawyer, situated in a belt from 7 to 8 miles in width. None of these are at present at work.§

* Christian Bechtler, jeweller by trade, who resided near Rutherfordton, N. C., was urged by residents in Rutherford and adjoining counties to coin the gold of that neighborhood, as transportation to the only mint then existing (Philadelphia, Pa.) was hazardous and difficult. He commenced coining in 1831, and continued until his death, in 1843, when his nephew, C. Bechtler, Jr., continued the minting until 1857. No regular entries of the quantity of gold minted were made; sometimes as much as $4000 to $5000 were coined in a week; and for a period of ten years the annual quantity was fairly equal. See *Second Annual Report on the Survey of South Carolina*, 1857, O. M. Lieber, p. 135.

† In the Census Report for 1880, vol. xiii., can be found statistics for the Southern States, tabulated under the following headings: Directory of deep mines; Means of handling water in deep mines; Cost of supplies in deep mines; Directory of ditches; Cost of ditch-plant; Grades and dimensions of ditches; Length of water season; Placer-directory; Stamp batteries; Tunnels in placer-mines; Amalgamating-mills; Arrastras; and Roasting-furnaces.

‡ Unless otherwise stated, the mines are not at present working. The values of the ores are not given on our authority; the same is true of the dimensions of the ore-bodies in abandoned mines, and in such as could not be examined.

§ The best description of these mines has been given by Prof. S. F. Emmons,

The last work done in Maryland, which was merely exploratory, was during the winter and early spring of 1895, at the Bethesda mine, 7 miles northwest of Washington, by the Bethesda Mining Company. Some $20,000 to $30,000 had formerly been taken from a rich chimney in a 6-foot quartz-vein at this mine. The old shaft was continued to a depth of 102 feet, and the ore-shoot found to pinch out. Assays from the lower end of the chimney ran about $4 per ton. There are no sulphurets to speak of in the ore at this depth. The country-rock (micaceous schist) is slightly auriferous in places. It is stated that large areas of the saprolites will yield 18 cents per cubic yard. Sufficient water-supply for hydraulicking or sluicing is difficult to obtain.

Virginia.

The principal gold-region of this State is that lying in the Virginia belt (see p. 665). A small isolated belt is situated on the west side of the Blue Ridge in Montgomery and Floyd counties, where some petty mining has been done. Very little information could be obtained regarding the working mines of this State. In Louisa county, two were visited, the Luce and Slate Hill; at other points prospecting is going on, and preparations for mining are being made. So far as is known, none of the mines are at present working on an actively producing scale. The mines of Fauquier, Stafford, Culpepper, Orange and Spottsylvania counties are grouped around the junction of the Rappahannock and Rapidan rivers, in a belt some 15 miles wide.

Fauquier County.—The Franklin, Wycoff and Leopold mines are situated in the southern part of the county, near Morrisville.

Stafford County.—The principal localities are in the western part of this county, near the Rappahannock river. The Eagle mine, situated 12 miles northwest of Fredericksburg, was worked by the Rappahannock Gold-Mining Company in 1894; greatest depth, 250 feet; length of working, 600 feet.

The Monroe mine adjoins the Eagle on the northwest.

The Lee mine is situated near the Eagle.

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in a paper entitled "Notes on the Gold-Deposits of Montgomery County, Maryland," *Trans.*, xviii., 391 to 411.

The Rattle Snake mine adjoins the Eagle on the northwest. It was worked as a gulch-placer; large amounts of nuggets are reported, from ½ to 5 dwts.; some as high as 125 dwts.

Culpepper County.—The Culpepper mine is situated 18 miles west of Fredericksburg, on the Rapidan river. Prof. Silliman gives the average value of this ore as about $25 per ton, and the mining and milling expenses at $7 per ton (report made in 1836). Other mines in the district are the Richardville and the Ellis.

The Powhatan Land and Mining Company was working a mine in this county in 1894, treating the ore in Crawford mills and a new 10-stamp mill (Fraser and Chalmers), and was also putting in Frue vanners.

Spottsylvania County.—The oldest mines worked in this county were those operated by the United States Mining Company near the Rappahannock river, in the extreme northwest corner of the county. A description of the method of milling at this mine in 1835 is given on p. 682. At this time a 2-foot vein was open by adits and several shafts, the deepest of which was 80 feet. The average value of the ore is given in this early report as $25; cost of milling at 80 cents per ton. Other old mines near this property are the Marshall and the Gardiner. In the central portion of the county there is another group of mines, most prominent among which is the Whitehall. This mine was in active operation as late as 1884, and is said to have been lately reopened. Other properties are the Kiggins, Johnston, Pullian and Grindstone Hill mines; still further south are the Mitchell and Goodwin mines. They are located on Pigeon Run, along which considerable placer work was done in the earlier days. Both have been worked within the last ten years, but no paying vein was developed.

Orange County.—The gold-mines of this county are situated in the northeast corner, near the Rapidan river. The most prominent one among them is the Vaucluse, which was first opened in 1832. A description of the milling-practice at this mine in 1847 is given on p. 683. Other mines in the vicinity are the Orange Grove, Greenwood and Melville.

Louisa County.—The auriferous rocks run across the centra portion of this county in a southwesterly direction, forming a belt but a few miles in width. In this belt, near the center

of the county, and from 2 to 6 miles northeast of Mineral City (Tolersville), are the Louisa county pyrite mines. These large bodies of sulphurets, occurring in lenses with maximum thickness of 60 feet, and developed to a depth of over 600 feet, are probably of contemporaneous origin with the gold-veins. They show the same strike (north 30° east), dip (60° to the southeast), and pitch of shoots or ore-lenses (45° to the northeast) as the quartz-veins in the vicinity. Traces of gold are found in the pyrite deposits, and small gold-bearing quartz-veins have been encountered in the mines.* The scope of this paper will not permit a detailed description of these interesting deposits, or the method of mining and concentration pursued here.† It may, however, be of value to give the cost of labor at these mines, as this would apply equally to auriferous exploitations. The daily wages paid are:

	Dollars.
Carpenters,	1.25 to 1.75
Engineers,	1.40
Blacksmiths,	1.30 to 1.75
Drill-runners,	1.35
Helpers,	1.20
General labor (underground),	1.00
" " (above ground),	90 cts. to 1.00

The Tinder Flats placer-deposits are situated at the northern end of the pyrite-bodies, on both banks of North Contrary creek. This bottom was, perhaps, the best known and most productive source of placer-gold in the early days of Virginia gold-mining. At present, it presents the problem of reworking shallow placer-bottoms on a larger scale; and at the time of our visit, Mr. W. H. Case, of Charlotte, N. C., was testing the ground with this object in view. Water under natural head cannot be

* We quote from a letter written by Mr W. H. Adams, to whose kindness we are much indebted: "It is true that in the pyrites vein, as now opened, there are traces of gold and silver, but I do not think the average so high as $1 per ton, and have only found that gold carries in certain lines, and that nearly all the vein-matter is bare. There are in all of the properties easily traceable quartz-veins in the hanging and foot slates, which are gold-bearing to the extent of $4 to $15 per ton, but these veins are always narrow—about as you saw them in our No. 3 shaft" (3 to 7 inches). "They are, however, persistent, and I have no doubt that chimneys are to be found at points of contact on the veins and dikes, which chimneys will be found to be the source of much of the gold so prevalent in the streams of the neighborhood."

† For a description of the deposits, see "Origin of the Iron Pyrites Deposits in Louisa County, Virginia," by Frank L. Nason, *E. and M. J.*, May 5, 1894.

obtained here, the surrounding country being a but slightly indented pene-plain; it would probably have to be pumped from the North Anna river.

One half-mile southwest of the pyrite-deposits, on the same strike, is the Walton mine. This mine has produced some very rich ore from a pocket developed to a depth of 150 feet. The property has been tied up in litigation for the last ten years.

Near Tolersville, a vein known as the Fisher lode, striking parallel to the pyrite-veins, about two miles further east, has been opened by the Harris, Luce, Slate Hill, Louisa, and Warren Hill mines. Two of these were in operation at the time of visit.

The Luce mine has been developed to a depth of 220 feet, and drifts along the vein have been opened to a length of over 1000 feet. There are three parallel veins, the one worked on at present having a thickness of from 3 to 8 feet. A 20-stamp mill (Fraser and Chalmers), fed by hand, was in operation on the property.

The Slate Hill mine was first opened in 1850, and for a time was extensively worked. It is the southwest extension of the Luce, which formerly constituted a portion of the property. Two veins are developed to a depth of 150 feet. In a report made in 1853, the average value of the ore is given at $4 per ton; the cost of mining and milling at $1.40 per ton. The present company began operations in March, 1895. A new Huntington mill-plant has been erected, and the mine is being developed at lower levels.

Fluvanna and Goochland Counties.—The same narrow belt runs along the boundary of Goochland and Fluvanna counties, crossing the James river at Bremo Bluffs, into Buckingham county. No work but petty placer-mining is carried on in these counties at present; although from 1830 to 1860 they were the field of extensive operations. According to older reports, the surface-ore at some of these mines ran exceedingly high. In 1836, Professor Silliman gives the average value at the Busby mine as $160 per ton; at the Moss mine, as $140 a ton; and at the Fisher mine, as $60 per ton. Other well-known and, at one time, extensively-worked mines, are the Tellurium (stated to have produced $1,000,000), the Bertha

and Edith, Manning, Tagus, Gilmore, Collins, Walters, and Belzora.

Buckingham County.—This is the most southwesterly county of the Virginia gold-belt in which mines have been actively operated; although the occurrence of gold is reported in Appomattox, Prince Edward's, Charlotte, and Halifax counties.

The Booker mine, near Whitehall Station, was worked before the war by an English company. The deepest shaft is 110 feet. The ore was crushed in a Howland pulverizer, and milled $13 per ton.

Another English company operated the London mine, seven miles north of the Booker, for a number of years. Other mines of equal importance in their day are the Garnett and Mosley, Buckingham, Morton and Morrow.

On the west slope of the Blue Ridge, mining on a small scale, mostly placer, has been carried on along Brush and Laurel creeks, in Floyd and Montgomery counties. In the latter county, the Walters and Gardner mine was operated in 1893.

Gold also occurs in Patrick, Carroll, and Grayson counties; but probably only to a very limited extent, associated with copper-ores.

North Carolina.

The North Carolina mines are distributed in three main belts—the Eastern Carolina, the Carolina, and the South Mountain belts (see pp. 666–673).*

The mining-districts of North Carolina have been more extensively developed than those in any other portion of the South; although to-day a comparatively small number of the mines are in operation. Of these, very few can be said to be steady producers, most of the work being prospecting and preliminary development, with irregular and spasmodic output. Petty mining, chiefly in the placer-ground, is carried on by tributers in various parts of the State.

* For a fuller description of some of the mines, the reader is referred to:

Geological Report of the Midland Counties of North Carolina, by Ebenezer Emmons. New York. 1856.

"The Ores of North Carolina," by W. C. Kerr and George B. Hanna. *Geological Survey of North Carolina.* 1887.

"The Gold-Ores of North Carolina," by H. B. C. Nitze. *Geological Survey of North Carolina.* 1895. Bull. No. 3.

The Eastern Carolina Belt.—The principal mines are situated in Warren, Halifax, Franklin, and Nash counties, in an area covering about 300 square miles, and extending in a southwesterly direction from a point near the Thomas mine, 1½ miles northeast of Ransom's Bridge, to the Tar river.

Among the mines in this belt are the Thomas, Kearney, Taylor, Mann, Davis, Nick-Arrington, Mann-Arrington and Portis. Among these the two latter are, perhaps, of most importance.

The Mann-Arrington mine is situated in the northwest corner of Nash county, at Argo P. O. The country-rock is chlorite-schist, in part porphyritic, striking N. 60° E. and dipping 40° S. E. The ore-body consists of quartz lenses from minute size up to 12 inches in thickness, imperfectly bedded in the schists and often cutting the same at low angles. The quartz is usually saccharoidal. The mine has been opened to a depth of about 108 feet and, so far as is known, was last worked early in 1894.

The Portis mine is situated near Ransom's Bridge in the northeastern corner of Franklin county. The country-rock is diorite. The ore-bodies lie in two intersecting belts of reticulated quartz-veins, each about 9 feet in total width. No work further than prospecting has been done on these. Small irregular quartz-stringers occur promiscuously throughout the country-rock, and the saprolites in general are stated to be auriferous. The only work of any consequence done here was surface-sluicing and hydraulicking to a depth of 15 to 30 feet. Sufficient water-supply and head are difficult to obtain. It is stated that 1000 cubic yards, washed in one of the sluice-lines, yielded 1018 pennyweights of gold, the loose vein-rock obtained in this mass assaying about $8 per ton.

The Carolina Belt.—Alamance, Orange and Chatham counties are included in this belt, being its northern extremity; but little work of consequence has been done here.

Guilford County.—Among the principal mines are the Fisher Hill, Millis Hill, Hodges Hill (Hodgins), Fentress (North Carolina), Twin, Gardner Hill, Jack's Hill, North State (McCullough), Lindsay, Deep River, Beason, Harland and Beard, situated from 3 to 10 miles south and southwest from Greensboro

in a general direction towards Jamestown. The country-rock is granite.

The Fisher Hill and Millis Hill mines are six miles south of Greensboro. There are two systems of parallel veins, the first running north and south and the second northeast and southwest. The aggregate length of the veins on this property is stated to be 8 or 10 miles. The vein which has been most extensively worked varies from 10 inches to 4 feet in thickness and has been successfully operated at several points. The mill consists of ten stamps and was running in 1886 and 1887.

The Hodges Hill (Hodgins) mine is two miles east of the Fisher Hill. The ore is quartz and chalcopyrite in a flat vein from 6 inches to 12 feet thick.

The North Carolina (Fentress) mine is 9 to 10 miles south of Greensboro. The general strike of the vein is N. 25° E.; its dip ranges from 38° to 60°. The quartz outcrop has been traced for three miles. The ore is chalcopyrite in quartz, containing gold. It was in 1856 worked for copper; shipments yielding 14 to 23 per cent. At the lowest depth, 310 feet, the ore-shoot was 80 to 90 feet long and 34 inches wide. The thickness of the vein varies from this to as high as 13 feet.

The Twin mine is six miles southwest of Greensboro. There are two parallel veins separated by 4 feet of slate. The strike is N. 40° E. and the dip S. E. The thickness of the vein is about 18 inches, the ore being auriferous quartz, carrying chalcopyrite.

The Gardner Hill mine is three miles northeast of Jamestown. There are supposed to be three veins on the property. The main vein strikes N. 20° E. and dips westward. Its thickness is from a few inches to 3 feet. The vein-matter is auriferous quartz, carrying chalcopyrite and some pyrite, inclosed in granite, with a slaty gouge on each side of the veins. The mine is 110 feet deep, and was said by Emmons (1856) to have yielded $100,000 from $10 to $20 ore. Tentative assays of the dumps, estimated at 25,000 tons, are said to show $3 to $10 per ton.

The North State (McCullough) mine is situated two miles east of south from Jamestown. The vein-fissure strikes northeast and dips 45° to 80° S. E. The mine was opened to a depth of 325 feet, where the vein was 4 to 8 feet thick. At

the surface it was 2 feet; at the 60-foot level, 4; at the 90-foot level, 10; at the 130-foot level, 24; and at 325 feet (the last work) 4 to 8 feet thick. The ore is quartz, carrying gold, pyrite and chalcopyrite. The brown ores extend to a depth of 130 feet and are said to have yielded from $1.50 to $5 per bushel ($15 to $50 per ton). The equipment, 20 stamps, etc., was last operated in 1884.

The Jack's Hill is on the northern and the Lindsay on the southern extension of the North State vein.

Randolph County.—The mines are in the central and western part of the county. The country-rock is argillaceous and chloritic schist, probably in large part sheared eruptives. At the Hoover Hill the rock is a massive porphyrite.

The Sawyer mine is 5 miles northwest and the Winningham, Slack, Winslow and Davis Mountain mines are from 2 to 5 miles southwest of Ashboro.

The Hoover Hill mine is situated about 10 miles west of Ashboro and 17 miles east of south from High Point. The country-rock is a basic eruptive which is partially brecciated, the included fragments being hornstone. In part the rock is slightly schistose. The ore-bodies consist of belts in this porphyrite, which are pyritic and filled with reticulated quartz-veins from less than 1 inch to 12 inches in thickness. The strike of the belts is N. E. and the dip 30°–60° S. E. The ore-bodies are intersected by pyroxenic dikes. The mine has been opened to a depth exceeding 300 feet. The so-called Briols shoot at this depth furnished ore worth $8 to $10 per ton. The mine was working in June, 1895.

The Wilson-Kindley mine is one-half mile southwest from the Hoover Hill, and the formation is similar.

The Jones (Keystone) mine is 18 miles east-southeast from Lexington. The country-rock is a very schistose phase of the brecciated porphyrite described at Hoover Hill. The strike is N. 45° E. and the dip 80° N.W. The ore-bodies consist of separate belts, 12 to 15 feet wide, of the schists, impregnated with auriferous pyrites and quartz-stringers. The entire width of the ore-bearing ground is stated to be 50 to 110 feet. The ore is cheaply mined in open cuts by quarrying. A 10-stamp mill stands on the property. The ore is stated to mill $2. Assay value $2 to $7 per ton. Pan-concentrates run $22 per

ton. The mine is at present in operation. The Uharie river, 2 miles distant, is the nearest supply from which water could be furnished by pumping, for hydraulicking and sluicing purposes.

The Herring (Laughlin) and Delft mines are in the vicinity of the Jones.

The Parish mine is $\frac{3}{4}$ of a mile southwest of the Jones. The country is similar. The ore is free gold in association with actinolite.

The Uharie mine is near the Montgomery county-line on the Uharie river. The ore-bodies are similar to those of the Russell, which is a short distance S.W. (see p. 700); but unlike that at the Russell, the work here has been underground.

Davidson County.—The Lalor (Allen), Loftin, and Eureka mines are situated from 2 to 3 miles southeast of Thomasville in the granite. The ores contain gold, silver and copper.

Two of the more important mines in this county are the Silver Hill and the Silver Valley.

The Silver Hill (Washington) mine is 10 miles southeast from Lexington. The country is chloritic schist striking N. 35° E. and dipping 57° N.W.; it is accompanied by an eruptive porphyrite similar to that of Hoover Hill. The ore is slate and quartz, carrying a complex mixture of pyrite, galena, zinc-blende and chalcopyrite. The galena is rich in silver. A general average of 200 assays of Silver Hill ore shows:*

	Per cent.
Galena,	21.9
Pyrite,	17.1
Chalcopyrite,	1.8
Zinc-blende,	59.2
Silver and gold,	0.025
	100.025

The mine has been opened to a depth of 660 feet. There are two parallel veins or lodes, known as the East and West, about 28 feet apart. The strike is N.E. and the dip 45° N.W. At the 60-foot level they come together, making 20 feet in width; at the 160-foot level the distance between the veins again widens to 32 feet, and the dip approaches the vertical.

* *Ores of North Carolina*, by W. C. Kerr and G. B. Hanna, 1887, p. 197.

At the 200-foot level the width of the west lode is 10 to 15 feet. This mine was discovered in 1838; it was last worked 12 years ago.

The Silver Valley mine is situated 5 miles northeast of the Silver Hill. The character of the country and the ore are similar to those at Silver Hill. The strike of the lode is N.E. with a dip of 45° N.W. The hanging is siliceous argillaceous schist, and the foot-wall, a hard horn-stone (devitrified quartz-porphyry). The outcrop is a barren milky quartz, 20 feet wide; the sulphurets come in at a depth of 60 feet. The mine has been opened to a depth of 120 feet. The lode is from 5 to 12 feet in width and consists of alternate bands of slate, quartz and sulphurets, the latter seams being from 3 to 18 inches thick. A 20-stamp mill stands on the property. The mine was last operated in the latter part of 1893, and the ores were smelted in a furnace at Thomasville. Many attempts have been made at various times to treat these complex ores, but unsuccessfully until this last time. A description of this smelting-process, by Dr. G. W. Lehmann, of Baltimore, Md., is therefore deemed of interest and is given here in his words:

"The smelting-plant, situated at Thomasville, N. C., on the line of the Southern Railroad and within 13 miles of the mines of the Silver Valley Mining Company, was erected especially for the treatment of the refractory ores from this mine.

"The composition of the ore is zinc-blende, galena, iron sulphides, together with some little copper, silver and gold. An average analysis representing a large lot delivered at the smelter gave: Zn, 28 per cent.; Pb, 12 per cent.; Cu, 0 5 per cent.; Ag, 21 ounces per ton; Au, 0.06 ounces per ton. Quite a number of patent processes have been in operation since the last 10 years at the works in order to profitably reduce the several metals, but none of these processes have gone beyond the experimental stage, since none of them proved a commercial success, until about two years ago. At that time Mr. Robert Memminger, of Newark, N. J., erected a plant which deals with the subject of treating refractory ores successfully. The plant consists essentially of:

"1. Down-draft jacket furnace connected with two horizontal jackets, one on each side of the furnace;

"2. Two condensers connecting with the horizontal jackets;

"3. Vat house with a series of large vats to receive the flow of liquor from the condensers and to collect the lead and zinc residues;

"4. A separate plant for the treatment of the lead residues;

"5. A separate plant for the treatment of the zinc residues.

"The down-draft furnace, as far as charging and general construction is concerned, is operated in a similar manner as any ordinary jacket-furnace, but the arrangement of the tuyeres is different, and the current of air from the blowers, necessary for the complete combustion of the refractory ore, is carried down through the charge; thence through the horizontal jackets, the condensers,

through two powerful suction-blowers along a series of dust-chambers, and out through the stack. A constant spray of water meeting the volatile metallic fumes of lead and zinc (together with what silver the zinc fumes carry along) in the two condensers, deposits all the metallic products, and carries them with the liquor into a series of vats where the lead sulphite or sulphate is deposited on the bottom of the vats, carrying the silver with it, whilst the zinc remains in solution and is precipitated out of this solution as zinc oxide.

"During the operation the slag is drawn off from openings near the bottom of the horizontal jackets near the furnace proper, whilst the matte is collected in the well of the furnace and tapped. This matte carries the copper, gold and most of the silver. It is necessary to prepare the charges to the furnace so as to have not less than 5 per cent. of copper in your charge; otherwise the resulting matte would be too low in copper and would have to be treated over and over again. Gold concentrates and even dry ores can be used with advantage as fluxes, and will help to make the process more profitable."

The cause of closing down the furnace was the difficulty of obtaining sufficient copper-ores for fluxing.

The Conrad Hill mine is situated 7 miles east of Lexington. The country rocks are siliceous, chloritic and argillaceous schists, striking N. 10°–20° E., and dipping 80° W. There are two systems of veins, one parallel to the schists, and the other cutting the strike of the schists. The vein-matter is quartz and siderite, carrying chalcopyrite and gold. The mine has been opened to a maximum depth of 105 feet.

Montgomery County.—The mines of this county are situated in the northern-central and northwestern parts, along the range of the Uharie mountains.

The Carter and Reynolds mines are some 6 miles northeast of Troy. They have been worked to a depth of 100 and 80 feet, respectively. Telluride of gold is stated to occur here.

On the northwest side of the Uharie mountains is a series of gravel-mines situated in a line between the mountains and the Uharie river. Among others may be mentioned the Bright, Ophir* (Davis), Spanish Oak Gap, Dry Hollow, Island Creek, Deep Flat, Pear Tree Hill, Tom's Creek, Bunnell Mountain, Dutchman's Creek, and the Worth. The available portions of these placers have been exhausted so far as the present supply of water will answer. The Beaver Dam placer is located about 5 miles west of Eldorado.

The Sam Christian mine is situated on the west side of the

* The saprolites have been explored here, and a belt 30 feet wide was found to mill $3 per ton.

Uharie mountains about 9 miles southwest of Troy. It was at one time extensively worked as a gravel-mine, the water being obtained by pumping from the Yadkin river about $2\frac{1}{2}$ miles distant. The two principal channels were the Dry Hollow and the Sam Christian cut. The thickness of the gravel varied between 1 and 3 feet. The gold was coarse, mostly in nuggets from 5 to 1000 dwts. The country-rock is the Monroe slate, accompanied by large masses of volcanic breccias and cherty felsites (devitrified quartz-porphyry) which contain small quartz fissure-veins from $\frac{1}{2}$ to 3 inches in thickness, striking N. 70° W. and dipping 60° N.E. Several shafts have been sunk on some of these narrow veins; but the attempts at deep mining were failures.

Most of the deep mines are situated in the extreme northwestern corner of the county, with Eldorado in their center.

The Russell mine (Glenbrook Mining Company), is about 3 miles northeast from Eldorado and but a short distance from the Montgomery county-line. The country-rocks are argillaceous-slates, both of soft and silicified types. Calcite occurs as a coating and in small veinlets. In part at least, if not altogether, these slates are sedimentary; the bedding and cleavage-planes usually coincide, though not always. The strike and dip is very variable. Diabase dikes occur in the country, but not in close proximity to the mine. The ore-bodies consist of parallel belts in the slates, impregnated with iron sulphurets (2 to 4 per cent.) and free gold, together with quartz stringers. There are at least six of these belts within a distance of 2000 feet across the strike. One of the largest is opened by the Big Cut, an open pit about 300 feet long by 150 feet wide by 60 feet deep. On the eastern edge of this cut is a shaft 150 feet deep, from the bottom of which the ore has been stoped upward. It is stated that the entire material from the cut averaged about $2 per ton, mill-yield. There were some rich streaks from 4 to 5 feet wide which went much higher. Two stamp-mills are situated on the property; the new mill contains 40 stamps and the old one (now in ruins) 30 stamps. It is proposed to treat the Russell ores by the cyanide-process; and the American Cyanide Gold and Silver Recovery Company, of Denver, Col., is making preparations for the installation of a plant.

The Appalachian (Coggins) mine is located near Eldorado.

It is similar in character to the Russell, showing large bodies of low-grade ores. The mill (40 stamps) was built in 1884.

The Morris Mountain mine is located one mile west of the Appalachian. The ore-bodies are similar to those at the Russell.

The Riggon Hill mine is located 3 miles east of Eldorado. The ore-body consists of a quartz-vein, $2\frac{1}{2}$ feet in thickness, lying in and with the slate country. It has been opened by a shaft 100 feet in depth. Some very high-grade ore (both in gold and silver) is reported from here. Prospecting-work was being carried on during the past summer.

The Steel mine is situated about 2 miles southeast of Eldorado. The country is silicified schist, striking N. 25° E. and dipping 70° N.W. The ore-bodies (9 to 12 feet in thickness) consist of the schists impregnated with sulphurets (galena, blende, chalcopyrite and pyrite) and intercalated with quartz stringers. The ore contains gold and silver and was formerly worked in a 40-stamp mill. The Saunders mine is an extension of the Steel.

The Moratock mine is situated 8 miles south of Eldorado. The country-rock is a massive, devitrified quartz-porphyry and volcanic breccia. It is very sparingly impregnated with sulphurets (pyrite and some chalcopyrite). Several small quartz fissure-veins (less than 1 inch in thickness) intersect the mass. The mine consists of a small quarry in the quartz-porphyry. A 10-stamp mill, equipped with a cyanide-plant, stands on the property, and was last in operation in July, 1893. The ore was reported to be of too low grade to be profitably treated.

Stanley County.—The mines are located in the northeastern portion of the county, more or less on the line of the Southern Railroad branch running from Salisbury to Norwood. Among the more important properties are the Haithcock, Hearne, Crawford, Lowder, Parker, Crowell and Barringer.

The Haithcock and Hearne mines are about two miles northwest of Albemarle. The country-rock is clay-slate, striking N.E., and associated with eruptives. The quartz-veins are stated to be from 2 to 6 feet in thickness.

The Crawford mine, situated 4 miles southeast from Albemarle, is a newly-discovered placer, and is described in detail on p. 728.

The Lowder mine is situated 4 miles west of Albemarle. It was opened in 1835, but has not been operated since the war. Previous to that time it was worked along the outcrop and to a depth of 65 feet. The quartz-vein is stated to be $3\frac{1}{2}$ feet in thickness, lying approximately with the slates in strike and dip. Recently the mine has been unwatered and some prospect-work has been carried on.

The Parker mine (The New London Estates, Capt. H. A. Judd, Mgr.) is situated at New London. The property comprises about 1200 acres. The country-slates resemble those of the Monroe type (see p. 669); they are intruded by successive flows of greenstone porphyry and more basic eruptives, in part brecciated. The mine-shafts have disclosed at least two volcanic sheets, from 2 to 3 feet thick each, lying horizontally and separated by sedimentary slates. In places the greenstone is squeezed into nearly vertical schistose masses. The country is intersected by numberless quartz-stringers, and several larger quartz-veins, which are auriferous. The principal work at the Parker consisted of hydraulicking in several old gravel-channels, which are stated to have yielded over $200,000. The gold was coarse, usually in nuggets from a few pennyweights up to 3 pounds. The fineness of the gold is 950 to 970. In one of the hydraulic cuts the bed-rock underlying the grit was decomposed greenstone. Test-pits have shown that this bed-rock is but a sheet of greenstone about 3 feet thick, and that it is underlain by another auriferous gravel-deposit, which may be considered virgin ground, as no attempt has yet been made to work it. There would be no great difficulty in getting a sluice on the bed-rock beneath this lower grit, with sufficient fall to carry off the tailings.

The hydraulicking-plant is very extensive. It consists of a Worthington compound duplex condensing-pump, with 2 100-horse-power boilers (using 7 cords of wood per day, at $1), situated at the Yadkin river, $4\frac{1}{4}$ miles from the stand-pipe at the mine, and 340 feet below the same. The pipe-line on the lower lift is 20 inches in diameter, flange-riveted, made of $\frac{3}{16}$-inch steel; on the upper lift is a similar iron pipe 12 inches in diameter. Expansion-joints are placed every quarter of a mile, and the full length of sleeve (8 inches) is necessary to take up

the maximum expansion and contraction of the pipe caused by changes of temperature. The capacity of the pump is 1,500,000 gallons in 12 hours; the head furnished from the top of the stand-pipe to the mine-workings is about 90 feet.

Besides the gravel-channels at the Parker, the saprolites are, in general, auriferous; and a combination sluicing- and milling-process (Dahlonega method, see p. 742) was at one time attempted here. The bank was undercut with powder and the shattered mass moved with the giants. The material ran about 50 cents a ton in the mill; but only a small percentage of it was quartz, and an attempt to select the latter proved unsuccessful. The tailings in the mill were reasonably low; but the losses of fine gold in the overflow from the mill-tank, in connection with the exhaustion of the richer available saprolites, led to the abandonment of the process.

The mill is a 10-stamp one, built by the Mecklenburg Iron Works, of Charlotte, N. C. The weight of the stamps is 650 pounds. In the Dahlonega practice 4 drops were given 80 times per minute, and round punched screens were used; no inside plates; about 50 per cent. of the gold was saved in the mortars between the dies. The total cost of milling (including 1 cord of wood, at $1, with 1 hand on each shift, at $1) was $4 per 24 hours.

The only work at present being done at the Parker is that of prospecting and developing some of the larger quartz-veins on the property. At the time of our visit the Ross shaft was 80 feet down, and in progress of sinking. At 250 feet it is calculated to strike the main vein, which is stated to be from 18 to 30 inches in thickness. The same vein had been exposed in a 130-foot shaft, to the west of the Ross, where assays of the quartz showed $3 at the 85-foot and $7 at the 130-foot-level. The dimensions of the Ross shaft are 5 feet 6 inches by 11 feet, inside measurements, with three compartments, the ladder-way being in the center. The timbers (10 by 12 inches white oak) are placed in square sets, with 5-feet centers. The cost of timber is $7 per thousand. From present indications, the cost of this shaft complete (including timbering) will be $10 per foot for the first hundred feet, $12 for the next hundred, and $15 for the last fifty feet.

Cost of labor:

Mine foreman (who also does the framing),	12-hr. shift,	$1.50
Helper to same,	" "	1.00
Blacksmith,	10-hr. "	1.00
Underground men,	12-hr. shift, 75 to 85 cents.	

The Crowell mine is situated near the Parker. The ore-body is a pyritic belt in the country-slate, from 4 to 7 feet in thickness, with a narrow pay-streak. The strike is N. 10° W., and the dip 45° N.W. The mine has been worked to a depth of 125 feet.

The Barringer mine is situated 4 miles southeast of Gold Hill. The gold is associated with limestone, and very rich ores are stated to occur here.

Moore County.—The mines are situated in the northern and northwestern parts of the county. The Jura-Trias sandstone, the eastern limit of the Carolina belt, passes in a southwesterly direction through the central part of the county, near Carthage.

The Bell mine is situated 8 miles north-northwest from Carthage. The country-rock is a garnetiferous chloritic schist, striking N. 55° E., and dipping 75° N.W. The ore-body consists of a 4-foot belt in the schists, containing a small percentage of finely disseminated pyrite and intercalations of siliceous seams from $\frac{1}{8}$ to 4 inches in thickness. The entire vein-matter is said to run $12 a ton. It is stated that the pay-streak, 4 to 8 inches thick, lay against the foot-wall, and that about 2 feet of the material on the foot-wall side was mined and milled, yielding as much as $30 a ton. The mine has been worked to a depth of 110 feet, and for a length of 800 feet.

The Grampusville mine is 3 miles southwest of the Bell.

The Burns mine is situated 11 miles west-northwest from Carthage, on Cabin creek. The country is sericitic and chloritic schist, in part silicified. The strike is N., 20° E., and the dip 55° N.W. The ore-bodies consist of certain belts of the country, impregnated with pyrite and quartz in lenticular stringers. The ore is mined in large open cuts, 20 to 100 feet wide and 50 feet deep. The average ore is stated to run from $2.50 to $3.00 per ton in free gold. In 1894, the ores were

being treated in five Crawford mills. No attempt had, up to that time, been made to concentrate and treat the sulphurets.

The Clegg, Cagle, Bat Roost, Shields, and Brown mines, are situated from $\frac{1}{2}$ to 3 miles west and north of the Burns. The character of the country-rock and of the ore-bodies is similar to that of the Burns.

Anson County.—A small patch of crystalline rocks, lying on the south side of the Jura-Trias sandstone, is gold-bearing. Two mines, the Hamilton (Bailey) and the Jesse Cox, are situated about 2 miles southwest of Wadesboro. They are unimportant, and are not working at present.

Rowan County.—The mines are located in the southeastern portion of the county, in three general groups:

1. In a line extending from 2 to 9 miles southwest of Salisbury, and 1 to 3 miles east of the North Carolina Railroad including the Hartman, Yadkin, Negus, Harrison, Hill, Southern Bell, Goodman, Randleman, and Roseman mines. Not sufficient is known of these to admit of any intelligent description.

2. Two to 7 miles east and southeast of Salisbury, in the Dunn's Mt. granite area, including the Dunn Mt., New Discovery, Bullion, and Reimer mines. Of these, the Reimer is fully described on p. 753, and will serve as a type for the others.

3. Nine to 10 miles southeast of Salisbury in the metamorphic schists, including the Gold Hill, Dutch Creek, Gold Knob, Holtshauser, Atlas, and Bame mines.

The Gold Hill district was at one time one of the most important mining centers in North Carolina, if not in the whole South, although at present no work of consequence is being carried on there. It is situated about 14 miles southeast of Salisbury, in the southeast corner of Rowan county, extending into Cabarrus county on the south and Stanley county on the east. The country-rocks are chloritic and argillaceous schists, striking north 25° to 30° E. and dipping 75° to 85° N. W. A diabase dike cuts the schists near the village of Gold Hill. The character of the ore-bodies is that common to these schists elsewhere, consisting of certain belts in the schists filled with pyritic impregnations and imperfectly conformable lenticular veins and stringers of quartz. The principal part of the gold-bearing zone is $1\frac{1}{2}$ miles long from northeast to southwest and

$\frac{3}{8}$ of a mile wide. There are many well-defined veins in the district, among which the more prominent ones are the Randolph, Barnhardt, Honeycut, Standard, Trautman and the McMackin. Some of these, such as the Trautman and McMackin, are heavy in argentiferous galena.

The first gold was discovered in 1842, and it is stated that in the 14 succeeding years the total production of the various mines was $2,000,000. In 1853 there was a population of about 2000 in the Gold Hill camp, at which time the Gold Hill Mining Company operated 5 Chilean mills and 40 to 50 rockers, working 300 hands. Between 1845 and 1850 the Randolph shaft was put down to a depth of 750 feet. This is the deepest gold-mine shaft in the South. The Randolph vein was worked in three principal lenticular ore-shoots, pitching to the northeast and varying from 50 to 200 feet in length and from a few inches to 6 feet in width. It is stated that remarkably rich ores were obtained in these days, large quantities yielding from $100 to $500 per ton in the mill.

The first stamp-mill (20 stamps) was erected in 1881, in which year the mine was unwatered to the depth of 400 feet. The last regular work was done in 1893 by the New Gold Hill Company, Mr. Richard Eames, manager, when the ores from the Barnhardt vein, which are high in copper, were milled in a 10-stamp mill. Mr. Eames carried on some laboratory-experiments in 1892 for a cyanide-treatment of the Gold Hill ores, and obtained an extraction of 60 per cent. on 100 pounds treated. During the past summer Mr. Bloomer, of London, has been experimenting with cyanide, but with what result is not known. Chlorination of the Gold Hill ores has been advised but never carried out.

At the present time the only work in Gold Hill is being done by tributers, who cart the decomposed material from the old mine-dumps to the Barnhardt mill, receiving about 50 per cent. of the yield. This material mills about $1.50 per ton. The pulp from the stamps flows directly over a line of blankets 24 inches wide, which are washed every 20 minutes in a tank; and the concentrates are treated in a series of hollowed log-rockers, 12 to 14 feet long, provided with quicksilver-riffles (see p. 680), the tailings flowing off into the creek.

At the Isenhour mine (Cabarrus county), $1\frac{1}{2}$ miles southwest

of Gold Hill, the ores from a 3-foot vein are being ground in a Howland pulverizer of 6 tons capacity per 24 hours. The pulp is run over blankets, the washings from which are treated in rockers, as at the Barnhardt mill, with a yield of about $2 from ores that assay from $5 to $7.

The Gold Knob mine is some 5 miles northwest of Gold Hill in the same general zone of schists. As many as 11 separate parallel ore-leads have been explored. Of these, the Holtshauser vein was again opened during the past summer.

The Dutch Creek mines are in the vicinity of Gold Knob. It is stated that there are 20 veins on the property, some of which are copper-bearing. The strike of the veins is generally northeast; but there is a second system striking more northerly and intersecting the first. The ores have been largely worked to the water-level, below which they are highly sulphuretted.

The Atlas and Bame mines are on the southwest extension of the Dutch Creek veins.

Cabarrus County.—The metamorphic schists occupy a narrow strip along the eastern edge of the county, in which are located a series of mines, which might be considered an extension of the Gold Hill zone. Such are the Widenhouse, Nugget (Biggers), Eva Furr, Allen Furr, Rocky River, Buffalo, Reed and Phœnix.

The other mines of the county are situated in the granitic rocks near Concord and to the southeast and south of Concord. Such are the Joel Reed, Montgomery, Quaker City, Tucker and Pioneer Mills mines.

The Nugget (Biggers) mine is situated 12 miles southeast of Concord, near Georgeville. The principal operations here during the past two years have been hydraulicking on a gravel-channel, similar to that at the Crawford mine in Stanley county. The gold is coarse, usually in nuggets. Quartz-veins carrying argentiferous galena have also been superficially explored.

The Rocky River mine is 10 miles southeast of Concord. The country is chloritic schist striking N. 20° E. and dipping 70° N. W. Several lenticular quartz-veins, lying more or less with the schists, have been explored. The quartz contains pyrite, galena, blende and chalcopyrite. During the past year Mr. Wayne Darlington, E.M., has been carrying on some prospecting-work on one of these in a shaft 130 feet deep, the total

length of the drifts being about 200 feet. In the 80-foot level the quartz was $2\frac{1}{2}$ to 3 feet thick; but it pinched out at 130 feet. Some of the ore was heavy in sulphurets and rich in gold. Cross-cuts have exposed parallel quartz-bodies. However, it appears that no regular quartz-vein can be depended on. The more or less silicified schists enclosing the quartz are impregnated with sulphurets and intercalated with small quartz-stringers, which, taken together, will make large bodies of low-grade ores. It is in such that the possible value of the mine must be looked for.

The Buffalo mine, 1 mile northeast of the Rocky river, presents similar conditions.

The Reed mine is $1\frac{3}{4}$ miles southeast of the Rocky river. It is the site of the first discovery of gold in North Carolina. In 1799 a 17-pound nugget was found, and in 1803 one weighing 28 pounds. The placer-ground was worked vigorously in former years and much nugget-gold taken out. During the past year work at this mine was revived, but it appears to have been simply of a prospecting character and short-lived. The chloritic schists are accompanied by a large body of greenstone, intersected by numerous quartz-veins varying in thickness from 4 inches to 3 feet. Some of these are gold-bearing, and were formerly worked by a shaft 120 feet in depth.

The Phœnix mine is situated 7 miles southeast of Concord. The country schists are accompanied by a large mass of diabase, in which the auriferous quartz-veins are confined. The main vein is the Phœnix, which was extensively and successfully worked under the management of Captain A. Thies, now of the Haile mine, S. C. Operations ceased here about 1889. The Phœnix vein strikes N. 70° E. and dips 80° N.W. It varies from 12 inches to 3 feet in thickness. The ore-shoot, which is 300 feet long and pitches to the northeast, has been worked out from the 100 to the 425-foot level. The shaft was sunk to 485 feet, but not drifted from. The vein in the shaft averages 30 inches; but the rich pay streak, lying on the hanging, is only from 2 to 3 inches thick. It is believed, however, that if the vein were drifted on at the 425-foot level the former ore-shoot would be reached again. Another ore-shoot, the Big Sulphur, is situated 300 feet southwest of the above, and has been worked to the 180-foot level.

Captain Thies's work was confined to the 300-foot shoot. The ore was quartz, carrying 3 to 60 per cent. sulphurets (pyrite, chalcopyrite and traces of galena). Barite and calcite occur in the gangue. The cost of mining was $4 per ton. Assays show from $1\frac{1}{2}$ to 3 per cent. of copper. The mill-yield was $10 per ton, besides which the sulphurets contained $7.50. The concentrates ran $30. Chlorination was first introduced here in about 1880. This was the Mears process, later developed into the Thies process. A full description of this with costs of working at the Phœnix mine has been given in a paper by Dr. William B. Phillips.*

Of the mines in the granite belt, some prospecting-work was being carried on during the past summer at the Joel Reed mine within the limits of the town of Concord.

The Pioneer Mills group of mines is situated 13 miles south of Concord. No work has been done here since the war. The granite is accompanied by large masses of basic eruptives.

Union County.—The mines are situated in the metamorphic slates in the western part of the county. Among the more important may be mentioned the Long, Moore, Stewart, Smart, Hemby, Lewis, Phifer, Davis, Bonnie Bell, and Howie mines.

The Long, Moore, Stewart and Smart are characterized by the presence of complex sulphurets (pyrite, galena, blende and sometimes chalcopyrite). At the Moore mine the gold is associated with calcite, which exists in a pay streak 4 inches thick on the hanging-wall of a 5-foot quartz-vein.

The Bonnie Bell (Washington) mine is situated 8 miles west of Monroe. The country is argillaceous schist silicified in varying degrees, striking N. 55° E. and dipping steeply N.W. The ore-deposit consists of pyritic and quartz impregnations in the schists. The width of the ore-bearing belt is stated to be 14 feet. It is intersected by a diabase dike. The mine was in operation during the fall of 1894. Ores assaying from $4 to $5 per ton were treated in a Chilean mill and four drag-mills, of 10 tons capacity per 24 hours; the pulp was discharged on amalgamated copper plates and thence to a Gilpin county bumping-table. The concentrates assayed $22 and the tailings 50 cents.

* "The Chlorination of Low-Grade Auriferous Sulphides," *Trans.*, xvii., pp. 313–322.

The Howie mine is 1 mile southwest of the Bonnie Bell. The ore-bearing slates are said to have a total width of 400 feet, within which there are as many as 8 so-called parallel veins, varying from 18 inches to 16 feet in thickness. Sulphurets are rare, the gold occurring mainly as fine films on the cleavage-planes of the more or less silicified slates. It is stated that the ore, when last mined, yielded $13 to $14 in the mill. The mine has been opened to a depth of 350 feet. Numerous diabase dikes intersect the ore-bodies, which are said to be richer in the vicinity of the dikes.

The Monroe slates in the vicinity of Monroe contain some narrow auriferous quartz-veins, but they are scarcely of economical importance, at least so far as present explorations have gone.

Mecklenburg County.—This has been one of the most important and active gold-mining counties of the State, although at present only one of the many mines is steadily at work and producing, namely, the Ferris.

The mines are distributed all over the county, around Charlotte as a center. Among the more important are the Davidson Hill (1 mile west of Charlotte), St. Catherine, Rudisil, Clark (2½ miles west of Charlotte), Stephen Wilson (9 miles west of Charlotte), Smith and Palmer, Howell, Parks (1 mile northeast of Charlotte), Taylor and Trotter (3 miles southwest of Charlotte), Brawley (4 miles west of Charlotte), Arlington (6 miles west of Charlotte), Capps, McGinn, Alexander (8 miles northwest of Charlotte), Dunn (10 miles northwest of Charlotte), Henderson (7 miles northeast of Charlotte), Ferris, Tredinick (7 miles southeast of Charlotte), Ray (9 miles southeast of Charlotte), Simpson (10 miles southeast of Charlotte), and Surface Hill (10 miles east of Charlotte).

The Rudisil mine is 1 mile south of Charlotte. In the upper part of the mine the country is a silicified, chloritic and argillaceous slate. At a depth of 200 feet this gives place to a crystalline eruptive rock. The ore-body consists of two parallel veins close together and separated by slate; they are said to vary in thickness from 2 to 6 feet. The strike is N. 30° E. and the dip 45° N.W. The mine has been worked to a maximum depth of 300 feet in three principal shoots, some of which furnished very rich though highly sulphuretted ores. The largest of these shoots had a maximum length of 100 feet and

a maximum thickness of 15 feet; it pitched towards the south, and was followed down to below the 300-foot but never found in the 350-foot level. No attempt at concentration and treatment of sulphurets was made.

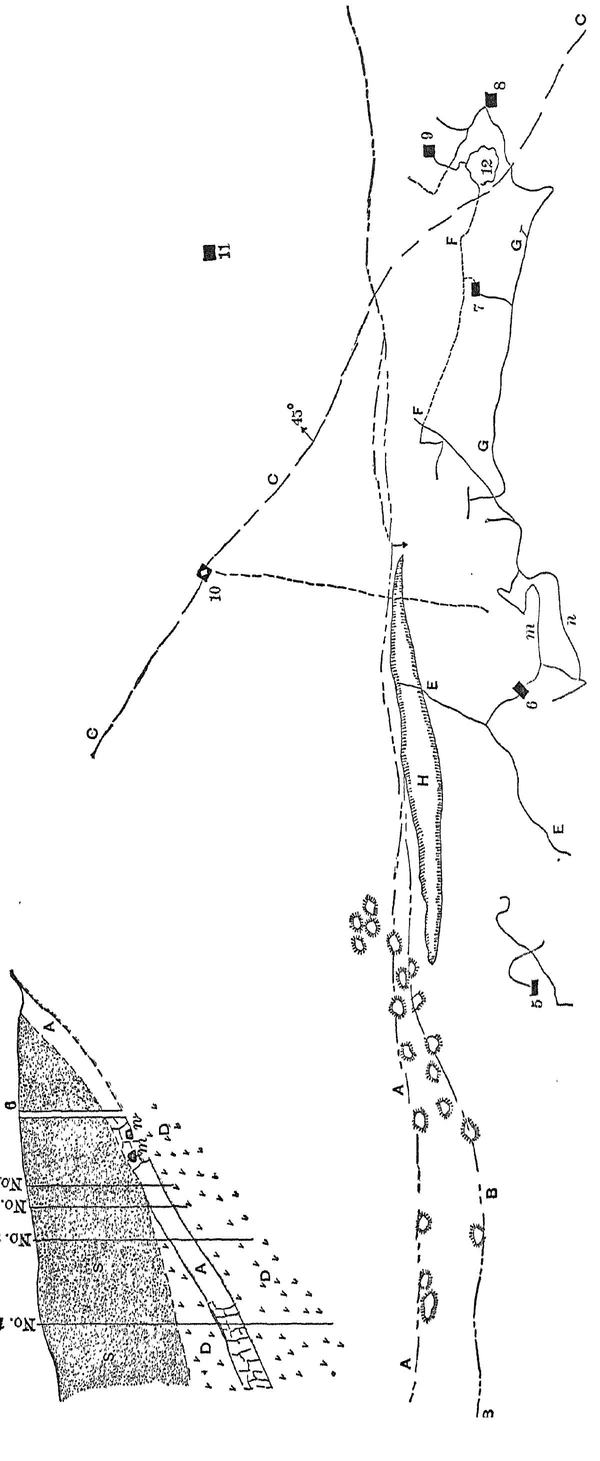
The Smith and Palmer and the Howell mines are supposed to be on the southwestern extension of the Rudisil.

The St. Catherine mine is on the northeastern extension of the Rudisil, and the general features are the same. The deepest workings are at the 370-foot level. It is reported that no large chimneys of solid high-grade ore were found below the 250-foot level; but between 200 and 370 a large shoot, 4 to 60 feet wide, of low-grade ore has been worked. The ores were treated by battery amalgamation, and the sulphurets were concentrated; these were probably shipped north or elsewhere for smelting.

The Capps mine is $5\frac{1}{2}$ miles northwest of Charlotte. There are two convergent veins, the Capps striking N. 30° W. and dipping 40° W., and the Jane striking N. 40°–60° E., and dipping steeply eastward (see Fig. 3). The actual intersection of the veins has not been found. The Capps was worked to a maximum depth of 130 feet in the Bissell shaft. The filling of the vein is quartz. Its thickness, as explored in the mine workings, was not less than 20 feet; definite walls were only found at a few points. The pay-ore was not uniformly distributed in the quartz, but generally occurred in layers. Four ore-shoots have been explored. The brown ores extend to a depth of 130 feet. The sulphurets are pyrite, with some chalcopyrite. The past production of the Capps has been estimated at over $1,250,000.

Between January and April, 1895, four diamond drill-holes (1-inch core) were bored on the Capps vein. Fig. 3 shows their position relative to the mine-workings in plan, as well as a vertical section of the ground which they explored. The Capps vein was penetrated by each bore-hole, and showed a regular thickness of about 20 feet, with walls of fine and coarse-grained diorite, at times porphyritic. The dip is quite constant, about 30° S.W. The vein-matter is quartz, averaging $6 to $7 per ton, as shown by assays of the drill-cores. These drill-holes are certainly very satisfactory, in so far as they prove the continuity in depth and regularity in thickness of the Capps vein; and on a large body of ore, such as this is, the assays of the drill-cores are of value as showing at least the presence of minable ores.

FIG. 3.



Plan of Capps Mine. Scale, 1 inch = 200 feet.

A, Capps vein; B, parallel vein; C, Jane vein; D, diorite; E, 90-foot level; F, 78-foot level; G, 130-foot level; H, open cut; S, saprolites; M, drift; N, drift; 1, borehole, 350 feet deep; 2, borehole, 250 feet deep; 3, borehole, 220 feet deep; 4, borehole, 200 feet deep; 5, Penman shaft, 80 feet deep; 6, Bissell shaft, 125 feet deep; 7, Mauney shaft, 130 feet deep; 8, Baldwin shaft, 120 feet deep; 9, Gooch shaft; 10, Old shaft; 11, Isabella shaft, 160 feet deep.

The McGinn mine comprises the Jane vein, worked to a depth of 160 feet in the Isabella shaft, and a cross-vein on the northern extension of the Jane, known as the Copper vein, which has been worked to the depth of 110 feet as a copper-mine.

The Ferris mine is situated 5½ miles northeast of Charlotte. The country is hydromica schist. The character of the vein matter is milky quartz, carrying free gold and pyrite. It lies with the schists, striking N. 25° E. and dipping 70° N.W. The quartz is broken up into stringers, the widest solid portion being 12 inches. The vein, as a whole, is stated to vary from 2½ to 5 feet in thickness. In one part of the mine it is intersected by a granite dike. In the fall of 1894 the mine was being worked by two shafts, respectively 56 and 95 feet deep. The ore was treated in a Chilean mill of 3 tons capacity. It is stated that the concentrates assay from $45 to $60 per ton.

Gaston County.—Among the mines of this county are the Oliver and Farrar (12 miles northwest of Charlotte), the Rhyne and Derr (17 miles west of Charlotte), the Duffie and Robinson (16 miles west of Charlotte), the Smith and Sam Beattie (13 miles west of Charlotte), the McLean (15 miles southwest of Charlotte), the Long Creek and the Kings Mountain.

The Long Creek mine is situated in the northern part of the county, about 6 miles northwest of Dallas. The country-rock is chloritic schist, striking northeast. There are three veins lying with the schists, and consisting of lenticular quartz-bodies. The Asbury vein was 6 to 8 feet thick, and contained rich shoots carrying sulphurets (pyrite, chalcopyrite, galena, blende and mispickel). A 10-stamp mill was running here in 1891, and in the following year a Crawford mill was put in, which was, however, soon abandoned, and the mine has since been practically idle.

The Kings Mountain (Catawba) mine is situated about 2

miles south of Kings Mountain, a station on the Southern Railroad, in the southwestern corner of the county. The country-rock is mica schist, striking N. 50° E., and dipping 70° N.W., intercalated with lenticular masses of siliceous magnesian limestone. They appear to be of sedimentary origin. The ore-bodies consist of large lenticular chimneys or shoots of this limestone, containing auriferous quartz and sulphurets (pyrite, chalcopyrite and galena up to 3 per cent.). Tellurides also occur in very small quantity. Five such lenses have been opened in the mine. In length they reach 100 feet and in thickness 20 feet, being separated by a black graphitic slate carrying coarse pyrite, which is, however, barren. The mine has been opened to a depth of 320 feet. At the time of our visit 40 tons of ore were being raised per 24 hours by a total force of 20 men. (Cost of mine-labor, 75 to 85 cents per day.) The rock is very tough, and 60-per-cent. dynamite is used for blasting. The mill-house is equipped with a well-constructed 30-stamp mill built by the Mecklenburg Iron Works, of Charlotte, and 5 Frue vanners (6 × 14 feet). Weight of stamp, 750 pounds. Twenty stamps were dropping—height of drop, 5 inches—71 times per minute. The ore was crushed through a 40-mesh brass wire screen. The mill-yield is stated to be $3 per ton, with a loss of $3 in tailings. Great difficulty was found in saving free gold, and the quicksilver gave trouble by flouring; this is ascribed to the graphitic slates which occur with the ore. The concentrates run $35 to $40. The total cost of mining and milling is $1.75. Two men are employed in the mill at $1 per day. The cost of wood is $1.35 per cord.

A plant for washing the surface brown-ores and saprolites is situated at the mine, and was, until lately, in successful operation. It consists of 2 sets of 12-foot log-washers. The slimes flowed over amalgamated copper-plates (12 feet by 5 feet), while the material carried up in the washer was screened through a ½-inch perforated revolving screen, and then through a 20-mesh brass wire revolving screen, from whence it passed over copper amalgamating-plates. The coarse material was taken to the stamp-mill. A large proportion of the gold remained in the log-washers; much was caught on the plates below the fine screens, and the smallest amount, which was all fine gold, was

caught on the slime-plates. Trouble was also experienced here by the flouring of the quicksilver. The bottom-land lying directly to the east of the mine is being worked in shallow pits by tributers, who wash the grit and soft bed-rock slates in sluice-boxes. Panning showed up very well here, and the ground might pay for hydraulic working on a large scale.

Lincoln, Catawba, Davie, Alexander and Yadkin Counties.—Gold has been found in these counties in isolated localities; but, with few exceptions, no mining work of any consequence has been done.

The Dixon mine, in Yadkin county, is a new discovery, and some prospecting was in progress during the past summer. The vein is reported to be several feet in thickness, of high-grade sugary quartz, containing some copper.

The South Mountain Belt, Caldwell County.—The Miller, Scott Hill, Pax Hill and Baker mines are situated within a distance of 1½ miles, on the waters of Johns river, near the southern boundary line of the county. The mines are located in each instance in close proximity to a wide dike of olivine diabase, which strikes through the country for many miles in a direction N. 20° W.

The Miller, Scott Hill and Pax Hill veins strike N. 50°–60° E. and dip N.W.; they are, so far as known, from 8 to 12 inches in thickness.

At the Baker mine the strike of the veins is N. 35°–45° W., and the dip is 60°–70° N.E. The thickness is from 2 to 5 feet; the ores contain auriferous and argentiferous galena.

The Bee Mountain mine is about 4 miles northeast of the Baker mine, and the ores contain zinc-blende, galena and chalcopyrite.

Burke, McDowell and Rutherford Counties.—By far the greater proportion of gold coming from these counties has been won by placer-mining. With few exceptions, the quartz-veins are too narrow to justify deep mining. But even in the cases where the veins are of sufficient width, mining operations have been very spasmodic and of limited extent. Placer-mining on a larger scale has been carried on during the past years only at a few points. Such are the Mills property and Hancock mines in Burke; Brackettown, Huntsville and Vein Mountain in McDowell, and Golden Valley in Rutherford county. None of

these, excepting the Mills place (Piedmont Mineral Co., Ltd.), are now regularly working. The latter is fully described on p. 732, and will serve as a type for the other mines of this district. Petty mining is almost constantly in progress in the above counties, as well as in certain parts of Cleveland and Polk counties on the south.

Of the quartz-mines those worthy of mention are the Idler, Elwood, and Vein Mountain.

The Idler (Alta or Monarch) mine is situated about 5 miles north of Rutherfordton, in Rutherford county. As many as 13 parallel quartz-veins have been explored here within a distance of ½ mile. The country is gneiss, striking about N. 60° W., and dipping 25° to 30° N.E. The veins strike N. 65° E. The vein-matter is quartz, containing sulphurets (pyrite and some chalcopyrite). The Alta vein has been explored to the depth of 105 feet; its thickness is from 10 to 22 inches; the ore is stated to yield $10 per ton. The mine has been worked in a desultory way, but is now under water.

The Elwood mine is 1½ miles southwest of the Idler. The character of the country, and of the veins, is similar to those of the Idler. The ore is reported to yield $5 in free gold. The mine was last operated in 1893.

The Vein Mountain mine is situated in McDowell county, on the Second Broad river. A series of as many as 33 parallel auriferous quartz-veins crosses Vein Mountain in a belt not over ¼ of a mile wide. The principal, and largest one of these, is the Nichols, which has been prospected in four shafts within a distance of 1200 feet, the deepest one being 117 feet. The strike of the vein is N. 80° E., and the dip 75° N.W. Its thickness is reported to vary from a few inches to 3 feet. The quartz is mineralized with pyrite, galena, blende, and chalcopyrite. The value of the ore varies from $2.50 to $70.00 per ton. There is a 10-stamp mill on the property, but it has never been operated on any regular output.

At Brackettown, 5 miles northeast of Vein Mountain, an expensive shaft has been sunk to a depth of 126 feet, on a parallel series of several narrow (1 to 6-inch) quartz-veins, with the fallacious hope that these would come together in depth. It is needless to say, that these small veins will not justify working alone, unless the intervening country (gneiss) is found to

contain auriferous sulphurets of sufficient richness to make large bodies of low-grade ores.

An isolated belt of gold-bearing rocks has been mentioned in Henderson county, N. C. (see p. 673). The only mine situated here is the Boylston, 12 miles west of Hendersonville. The country-rocks are fine-grained mica- and hornblende-gneisses and schists, in part much crumpled, striking N. 20°–30° E., and dipping 35°–60° N.W. The quartz-veins coincide, more or less, with the strike of the schists. The mine has been opened by a series of shallow shafts and short drifts on one of these veins, which is from 3 to 4 feet in thickness, with a pay-streak of 1 to 3 inches on the hanging; it is accompanied, in places, by a granitic dike. The ores are reported to average about $4 per ton (assay-value); sulphurets occur, chiefly pyrite and some chalcopyrite. A 10-stamp mill (in bad repair) stands on the property. It has not been in use since 1889.

In the northwestern corner of North Carolina, the copper-ores of some of the Ashe county mines, and some small galena-bearing veins in Watauga and Wilkes counties, are auriferous.

In the southwestern corner of the State, in Jackson, Swain, and Cherokee counties, some insignificant placer-mining operations have been carried on.

Gold is also stated to occur in Macon county, and this may be a northern extension of the Georgia belt (see p. 673).

Tennessee.

The gold produced in this State has been obtained entirely from petty placer-workings in Monroe, Polk, McMinn, and Blount counties. The most prolific sources have been the deposits along Coca creek, a tributary of the Hiwassee river in Monroe county. Other gold-bearing streams in this county are Citico and Cane creeks, and the headwaters of the Tellico river. Along Whippoorwill branch, a tributary of the latter, small gold quartz-veins have been discovered, but they have never been worked.

South Carolina.

The present gold-output of South Carolina is derived almost entirely from the Haile mine. There seems to be some prospect of the Brewer mine resuming operations, and there

are some minor explorations, mainly placer, throughout the State.

To show the extent and distribution of the gold-mining industry before the War, the following table comprising the working mines in 1859 is given:*

Chesterfield and Lancaster counties, . . .	21	working mines.
Spartanburg, Union, and York counties, . .	19	" "
Abbeville and Edgefield counties, . . .	10	" "
Greenville and Pickens counties, . . .	8	" placers.
Total in State, ...	58	

Some of these were probably minor operations, as Lieber† in his reports, made a few years earlier, complained of the lack of interest taken in the South Carolina gold-mines.

Carolina Belt, Chesterfield County.—The Brewer mine (see p. 762) is the main point of interest in this county.

In the same neighborhood are the old Kirkley, Leach and McInnis mines. Some gravel-mining has been done near the northern boundary of this county.

Lancaster County.—The Haile mine is fully described on p. 767. The Funderburk, 8 miles northeast of the Haile, and of the same character, was worked as late as 1887. The Clyburne property is situated 1½ miles southwest of the Haile. Some tributing is done here with rockers, on saprolites and gulch-deposits. Adjoining this on the southwest is the Gay mine, which shows ore-bodies of the Haile type, but is little developed. The most southerly occurrence of gold in this district is at the Williams mine, 7 miles southwest of the Haile.

York County.—There is no active work at present in this county. Among the older mines of this district are the Wilson, Wallace and Palmetto.

Union County.—About 3 miles south of Glen Springs are the West and the Thompson mines. Mr. Becker describes the veins as quartz-lenses similar to the Dahlonega type, interlaminated with mica- and hornblende-schists. The Thomson

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* *South Carolina Resources, etc.*, published by the State Board of Agriculture, Charleston, 1883.

† For a full discussion of the occurrence of gold and a description of the older mines in South Carolina, see O. M. Lieber, *Reports on the Survey of S. C.*, 1856, 1857 and 1858.

mine was operated last summer on a small scale, using the Dahlonega method of mining and milling.

Abbeville and Edgefield Counties.—Little information could be obtained regarding the mines of this district. The deposits are probably closely connected with, and of the same nature as those in McDuffie, Warren and Columbia counties, Ga. The Dorn mine, situated at the lower end of the Abbeville district, was opened in 1852. In the first year of its operation over $300,000 are said to have been taken from this mine; a yield of $100 per ton was considered a poor one. The rich pockets were, however, soon exhausted; and the mine, though worked again in 1866 with some reported success, has now been practically abandoned.

South Mountain Belt—Spartanburg, Greenville and Pickens Counties.—The gold district in these counties is probably a continuation of the South Mountain belt in North Carolina, and, as in that district, the gold produced has been obtained almost entirely from placer-deposits. Some of these are operated at present in a desultory manner. Among the more extensive ones might be mentioned the Wolfe creek and Tyger river placer deposits, located on the boundary between Spartanburg and Greenville counties at the foot of Hogback mountain. The gold in these bottoms is derived from small quartz-veins having the same strike and dip, and being in other respects similar to those of the South Mountain district in North Carolina. The gravel in the bottom is from a few inches to 5 feet in thickness. It consists of white saccharoidal and glassy barren quartz. In 1892, the Wolfe creek bottom was worked by the Wolfe and Tyger Mining Company, with a 2-inch nozzle giant, supplied with 45 feet head of water by a 4-mile ditch.

Georgia.

The principal gold-mining in this State is prosecuted along the Georgia belt proper, mainly in White, Lumpkin and Cherokee counties. Besides this, the J. Sep Smith mine, situated on the continuation of the Carolina belt in McDuffie county, has been worked for a considerable number of years at a profit.

Rabun County.—In this county some work has been done at the Smith's mine near Burton, and at the Moore Girls' mine, 12 miles northwest of Clayton.

Habersham County.—The only activity in the county is some development work a few miles northeast of Clarkesville.

White County.—The main mining district in this county is located near the picturesque Nacoochee valley and Yonah peak. Among the many Indian traditions of this neighborhood is that of extensive gold mining by the aborigines, but absolute proof of this is wanting.

The Lumsden mine is situated about 2 miles north of Nacoochee valley on Bean creek. Several rich quartz-stringers are here worked by tributers, using a combination of hydraulic and dry mining, the hard ore being hauled ¼ of a mile to a wooden 10-stamp mill driven by a 20-foot over-shot water-wheel. Five hands are stated to extract 70 to 80 dwts. per week by this crude method.

The Jarret mine adjoins the Lumsden on the south; a 20-stamp mill was operated here for some time by Mr. Childs of Athens, Ga., using the Dahlonega method (see p. 742). It has lain idle for the last seven or eight years.

The Yonah Land and Mining Company controls some 4800 acres of mining property situated mainly along the water-shed of Duke's creek. This company has pursued extensive vein explorations on their land under the direction of Mr. E. T. Whatley, formerly connected with the State Survey. This prospecting has disclosed a large number of auriferous veins, some of them with considerable continuity along the strike, which is about 20° N.E. They dip about 85° to the S.E., while the dip of the country-rock is slightly to the N.W. Although of low grade ($3 to $7 per ton) and small width (6 inches to 3 feet) some of these veins might, under close management, be mined and milled at a profit. The producing operations of this company are confined to placer-work with hydraulic elevator, in the bottom-land of Duke's creek. The elevator used and the method of work in the pits is similar to that pursued by the Chestatee Company (see p. 739). A 65-foot head of water is obtained from a 7-mile ditch line. The gravel averages about 3 feet in thickness, covered by 6 inches of peat and clay, and above this about 6 feet of soil over-lay. The gold consists to a large extent of extremely rounded and water-worn nuggets often aggregated in pockets. From one of these $1500 in nuggets was taken in two days.

Other properties in active operation in this district are the Longstreet placer, $2\frac{1}{2}$ miles northwest of Cleveland, and the Old Nacoochee and Hamby mines, operating 30 stamps under one management.

Besides these, there are quite a number of smaller operators, some working gravel in sluice-boxes, others mining rich seams in the saprolite and "beating" the ore in wooden stamp-mills, as for instance at the Thompson mine near the Yonah Land Company's property. Here the mining operations were carried on by a mother and son, the latter digging the quartz and carrying it on his back to the mill, while his mother attended to the beating.

About 6 miles northeast of Cleveland is the Loudville mine, at present not in operation.

The Loud mine is situated near Pleasant Retreat Post-office about 11 miles east of Dahlonega. At present, old gravel-piles are being worked here by a giant; water under 75-foot head is leased from the Hand-Barlow Company and is supplied by a ditch 25 miles long. Extensive cuts in the saprolites have been made here. An interesting feature of this mine is the occurrence of wire-gold.

Hall County.—Little work other than prospecting is going on here. Such is the case at the Merck and Currahee mines, near Gainesville. The Potosi mine, 12 miles northeast of Gainesville, has lately been reopened with the hope of finding the continuation in depth of a rich shoot once mined here. So far the search has been unsuccessful.

Lumpkin County.—The principal mining operations in this county are in the vicinity of Dahlonega, stretching from the Yahoola river, about 1 mile northeast of the town, in a continuous belt nearly 4 miles in width to the mining village of Auraria, a total length of about 6 miles. A general description of this belt and the method of mining and milling pursued here is given on p. 742. The following is a list of the more prominent mines at present (July, 1895) operated by the Dahlonega method in this district: Mary Henry (5 stamps); Hand (20 stamps); Singleton (10 stamps); Findley (40 stamps); Preacher Lot (10 stamps); Stanley (10 stamps); Hedwig (40 stamps), and Josephine (20 stamps).

At the Lockhart mine quartz from underground stopes is

treated in a 20-stamp mill (see p. 751). At the Garnet mine, about 6 miles northeast of Dahlonega, a plant is being installed for working dry ore; and some prospecting is being done in the old Battle Creek mine near Auraria, at one time renowned for its rich pockets. The operations of the Chestatee Company and that of the dredge-boats are described among the special cases below. Several of the above working mines are operated by lessees, the usual royalty paid being 25 per cent. for properties on which a mill and water-power is furnished, and 10 per cent. where these are absent. Among the more important idle mines operated at one time by combined hydraulicking and milling are the Ivy (60 stamps); Barlow (40 stamps); Rolston (20 stamps); Yahoolah (20 stamps); Fish-trap, London, New Gordon, Lawrence, Bast, Little Findley and Whim Hill mines. The nearest railroad-point to Dahlonega is Gainesville, 20 miles to the southeast. A connecting railroad branch between these points is looked for in the near future, and will greatly benefit the mining interests of this district.

Dawson County.—At present no active work is prosecuted in this county, excepting petty tributing. Among the mines formerly extensively operated by the Dahlonega method are the Cincinnati Consolidated, Etowah, Kin Mori and McGuire properties, all situated in the vicinity of the county seat, Dawsonville.

Forsyth County.—Small placer-workings represent the gold-mining of this county.

The Piedmont mines near Buford in the extreme north corner of Gwinnett county have been worked to some extent, and the occurrence of gold is reported in Milton and Fulton counties.

Cherokee County.—At Creighton near the eastern boundary of this county, is located the Franklin mine (See p. 757), which, together with the Haile mine of South Carolina, shows the brighter side of Southern gold-mining. Stimulated by the success of this mine, developments are being pushed on several other properties in this county, mainly along the approximate strike of the Franklin vein. The properties extend from the Dr. Charles* and Clippinger mines, about 3 miles north of the Franklin, in a more or less continuous line to the Sixes, Wilkinson, Cherokee, Georgiana, Cox and Worley mines

* This mine carries arsenical pyrites from the grass-roots down, and very little common pyrites.

in the southwestern portion of the county. Near the center of this belt, south of the Franklin, the Chester (formerly Latham) and the Strickland properties are being developed.

The same auriferous belt extends through a portion of Barton, Cobb and Paulding counties. In the latter county a high-grade quartz-vein has been opened up in the Yorkville mines.

In Douglas county, lenses of auriferous quartz were explored to some extent in former years, but no active mining is carried on at present.

This belt continues into Carroll county. Here a few mines are being developed in the vicinity of Villa Rica, among them the Clopton and Schoetbler properties; the latter has been producing bullion on a limited scale for the last few years.

Haralson County.—A few mines have been opened in the southwestern portion of this county, lying partly in the belt which, southwest of this point, is more extensively developed near Arbacoochee, Alabama. The most prominent among these mines is the Camille, near Tallapoosa, which was first opened in 1843. In 1887, the property was purchased by Messrs. Carpenter and Shaw, of Natchez, Mississippi, and a large amount of money was spent in developing the mine and erecting a stamp-mill and chlorination-plant. Work was however soon discontinued, the development of ore not proving satisfactory. Since then, several thorough examinations of the mine have been made, but a continuous operation has not resulted from them.

To the southeastward of this main Georgia belt gold is found in a few isolated districts; notable among these is its occurrence in Meriwether county. The only mine operated in this county is situated in the extreme northwest corner. It is known as the Wilkes mine, and is stated to have produced about $50,000 from 1873 to 1878, during which years the vein (composed of quartz lenses 8 to 10 inches thick) was mined to a depth of 130 feet. In the spring of the present year the mine was again opened, and the operations pursued here on a very limited scale by Mr. John Cross, lessee, are stated to be profitable. The ore, consisting of quartz with about 3 feet of the adjoining wall-rock, mills $4 per ton.

Carolina Belt.—In the eastern portion of the State an auriferous district, which probably represents the most south-

erly extremity of the Carolina belt, is developed to some extent in McDuffie, Warren, Wilkes, Lincoln and Columbia counties.

The most prominent mine in this district is the J. Sep Smith mine, in McDuffie county, which has been worked for a long time at a profit. This mine is situated 40 miles south of Thomson. A 3-foot vein of white quartz, striking nearly N. and S. and milling from $8 to $24 per ton has been developed by 2 shafts to a depth of 160 feet and for about 300 feet along the strike. The mill is a 10-stamp one, and is located 3 miles from the mine on Little river. No attempt is made to save the sulphurets. Tailings as high as $12 per ton are reported. The mine is worked on a very limited scale under the personal management of the widow of Col. J. Sep Smith, and is stated to have furnished to the family a comfortable and continuous income.

Other mines in this district are the Columbia, Egypt, Tatham and Williams in McDuffie county, the Warren in Warren county and the Magruder in Wilkes county.

Alabama.

During the summer of 1894 considerable attention was paid to gold-mining in Alabama. The following is a list of the mines in operation at that time, furnished by Dr. E. A. Smith, State Geologist: The Eckles, Wise, Lucky Joe, Crown Point, Red Rover, Lee, Arbacoochee and Sutherland mines, Cleburne county; the Pinetucky mine, Randolph county; the Bradford and Walker, Goldberg, Franklin and Idaho mines, Clay county; and the Blue Hill, Farrow, Eagle Creek, Johnson and Davis mines, Tallapoosa county.

Among this list there must have been a number of petty operations, as during the summer of 1895 no work more extensive than prospecting and developing was prosecuted in the State.

Cleburne County.—All of the more important mines of the county are located in the Arbacoochee district, situated 7 miles southeast of Heflin, the nearest railroad point. In the earlier days extensive placer mining was carried on about $\frac{3}{4}$ of a mile southwest of the mining village, Arbacoochee, principally in the Clear Creek valley. The auriferous deposit at this point covers nearly 100 acres, known as Sections 5, 6 and 7.

During the present summer a pocket of very rich quartz was opened up in one of the old placer-pits on the boundary-line between Sections 6 and 7. It is stated that between $1000 and $2000 of very coarse gold was taken from about 400 pounds of ore and the immediately overlying gravel. This find created considerable local stir, and prospecting was being pushed along the strike of the quartz-vein as far as the direction could be determined from the very limited dimensions of the ore-lens, the latter having a maximum width of 8 inches, a dip of about 30°, and pinching rapidly along the strike in a distance of about 6 feet. The ultimate value of this find will depend on the continuation of this shoot in length and depth, or the discovery of new ore-bodies along the strike of the veins.

The only hydraulic work in the State was carried on for a short time by the Arbacoochee Hydraulic Company on side-hill deposits, about ½ mile east of Arbacoochee. The limited supply of water and poor management are given as the reasons for failure.

At the Lucky Joe mine,* on Turkey Heaven mountain, an excellent stamp-mill (Fraser and Chalmers) was erected in 1893, and the mine was opened in a systematic manner. It was, however, abandoned in the following summer, the development of ore in the mine not proving satisfactory.

Other mines in this district operated more or less extensively at one time are the Annie Howe, Price, Red Rover, Eckles, Lee, Crown Point, Sutherland, Middle Brook, Bennie Field, Ballinger, Gold Eagle and Moss Back. On several of these properties there are well-constructed mills.

About 8 miles west of Arbacoochee, near Chulafinnee, prospecting has disclosed some rich quartz-stringers on the property of Mr. Burrel Higginbotham. Near it is the King mine, abandoned on account of sulphurets.

Randolph County.—The only mine of prominence in the county is the Pinetucky, probably the best known property in the State. It might be classed as belonging to the Arbacoochee district, and is located about 2 miles south of Micaville and 14 miles from Heflin, near the northern boundary of the county. This property was first discovered by Mr. Knight in

* For full description of property see *E. & M. J.*, vol. lvi., pp. 79 to 80, July 22, 1893, W. M. Brewer.

the early days of gold-digging. He is reported to have taken about $30,000 of gold from the vein. Numerous shallow workings, perhaps the most extensive seen in the South, running in a continuous line for more than half a mile along the outcrop, gave testimony of the extensive and uniform character of the vein. The strike of the vein is nearly north and south, the dip 20° E. and the average thickness from 8 to 9 inches. The quartz is highly vitreous, and bluish in color. The average value of the ore is stated to be nearly $40 per ton, assays of larger lots having shown as high as $150 per ton. About one-half of the gold is free-milling, the other half being contained in the sulphurets. The percentage of the latter is small; assays of concentrates are reported as high as $300 to $400.

The country-rock is a very garnetiferous hornblende-schist, in which the vein lies more or less conformably. Diamond prospect-drilling in the foot-wall has proved the existence of a large mass of light-colored granite at a depth of about 60 feet, which seems to have either cut off or changed the course of the vein in depth. The old workings have been carried to a depth of 70 feet, with drifts below the whole length of the outcrop. A few years ago a complete and well-constructed 10-stamp mill of Western pattern (Fraser and Chalmers) was erected on the property about 700 feet east of the outcrop. A vertical shaft was started in the mill-house, with the object of striking the vein in depth, and hoisting the ore direct to the grizzly and crusher situated at the top of the building. This shaft was continued to a depth of 50 feet, and then abandoned for lack of funds. In the spring of 1895 the property was leased to the Fair Mining Company, who began operations by sinking 3 vertical drill-holes. The first of these was sunk to a depth of 200 feet in the bottom of the mill-house shaft, but no vein was encountered. The second of these, sunk about 150 feet east of the old workings and to a depth of 180 feet, also failed to reach the vein. In the third, only 80 feet from the old workings, the vein (1 foot thick) was finally found at a depth of 70 feet. In gold-quartz veins of this size the result obtained by diamond-drill borings might often be misleading, as the gold-bearing vein can at times be distinguished from other quartz only by its gold-contents; about this the drill-core, and still more the cuttings used as assay-samples, can give no reliable

information. However, such explorations may disclose other facts of interest, as, for instance, in this case the discovery of granite overlying the vein in depth, which may give a clue to the formation of the vein and more intelligently direct search for it.

The prospecting-work at this mine was done with a small Sullivan drill ($\frac{7}{8}$-inch core). The drill-runner furnished by the Sullivan Diamond Drill Company, of Chicago, received $90 per month. The cost of underground labor in this district is $1 per day and for top labor 80 cents to $1; cord-wood, 75 cents per cord; freight to Heflin (by wagon), 14 miles, 20 cents per 100 pounds.

Near the center of this county, at Wedowee, some placers have been operated.

The Goldberg district lies in the extreme western part of the county, running partially into Clay county near Abner. Attention has been paid in this direction almost entirely to placer-mining along the bottom of Crooked creek.

Clay County.—The more important mining operations in this county have been carried on in the Idaho district. The last work done here was at the Idaho and California mines, at the latter of which a 10-stamp mill was operated. Other properties are the Chincopino, Franklin, Horn and Laurell mines.

Talladega County.—The occurrence of gold in this county is limited to the extreme eastern portion, the Cambrian and Lower Silurian formations occupying the remainder of the territory. The Riddle and Story mines have been worked to some extent.

The occurrence of gold in Coosa, Chilton, Chambers and Tallapoosa counties has been fully described by Dr. W. B. Phillips.* The latter county was at one time the seat of extensive mining operations in the Goldville, Hog Mountain, Silver Hill, Gregory Hill, Blue Hill and Farrow Mountain districts.

Some excitement was occasioned in the spring of this year by the discovery of gold along the Tennessee river, in Marshall county. Mr. Henry McCalley, of the Alabama Geological Survey, reports it to be a placer-deposit of small economic importance, the only possible natural derivation of which could

* Bulletin No. 3, *Geological Survey of Alabama*, 1892.

be from the metamorphic schists some 300 miles east by waterway.

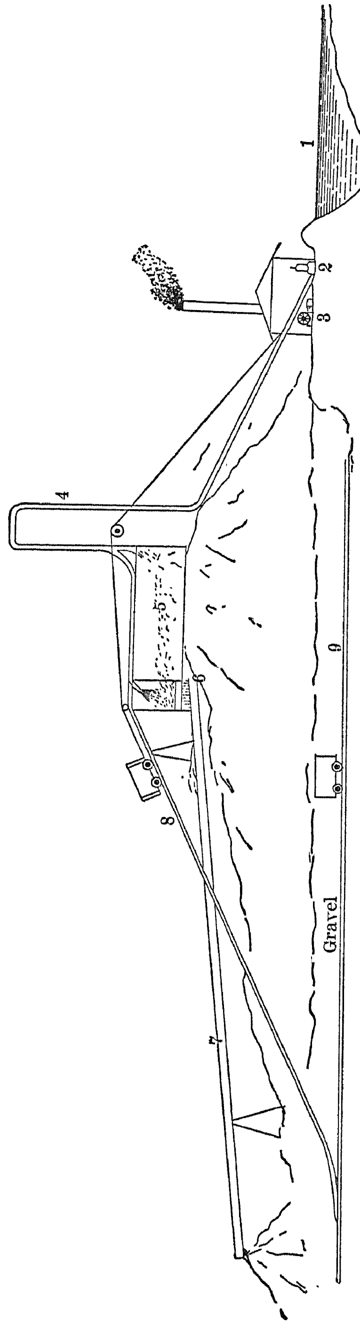
VI. Description of the Mining, Milling and Metallurgical Practice at some of the Specially Characteristic Mines.

The Crawford (or Ingram) Mine, Stanley County, N. C.

This mine is situated 4 miles southeast of Albemarle, in the Carolina belt. It represents a type of working in virgin placer-ground, the gold being coarse, usually in nuggets. The mining tract (180 acres) comprises a flat hollow or depression, averaging 250 feet in width, which is drained by a small branch. The country-rock is the dark greenish Monroe slate (sedimentary), lying in a flat synclinal trough. The auriferous grit, lying on the slate floor, is composed of angular fragments of quartz and country-rock bound in a clay matrix; the cement is often hard and stained a brownish or black color. The quartz is of a milky, vitreous variety, seldom showing ferruginous stains; some pieces show parallel walls (vein-structure) from a few inches up to 1 foot in thickness. No free gold has been found in this quartz. The thickness of the grit in the center of the synclinal basin is from 1½ to 2 feet, and of the over-lay 2 to 4 feet, thinning out towards the edges. The length of the deposit on the company's property is about a quarter of a mile. The adjoining property on the north is owned by Mr. F. A. Fesperman, whose place has been worked by tributers. The gold found at the Crawford is altogether coarse, from the size of a pin's head to nuggets of considerable weight. The largest nugget was found on August 22, 1895, and weighed 10 pounds. The so-called De Berry nugget, found April 8, 1895, weighed 8 pounds 5 ounces. These nuggets are scarcely at all water-worn, being rough and irregular in shape. The fineness of the gold varies from 850 to 900.

On the hill-side to the west of the placer-mine several quartz-veins have been explored by shallow openings along the outcrop. One of these is from 2 to 3 feet thick, and dips steeply to the east, cutting the slates both in strike and dip. The quartz, so far as explored, has been found generally barren, though in several places gold has been panned from the crushed rock; but no larger pieces have been found giving any possible clue to the nuggets of the placer-deposits.

FIG. 4.



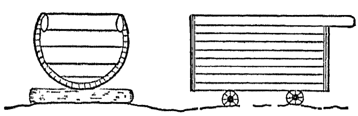
Sketch of Method of Working Gravel at Crawford Mine, Stanley County, N. C. (Not drawn to scale.) 1, reservoir ; 2, pump ; 3, hoisting engine ; 4, stand-pipe ; 5, washing-box ; 6, grizzly ; 7, sluices ; 8, inclined plane ; 9, track on bed-rock.

Gold was first discovered in this bottom in August, 1892, the property being at that time a portion of the W. S. Ingram farm. For two years it was worked spasmodically by tributers, and 16 to 17 pounds of nuggets were obtained. In 1894 the property was bought by the Crawford Mining Company of New York, and is now under the management of Mr. Richard Eames, Jr., of Salisbury, N. C. A sketch of the method pursued by this company is given in Fig. 4.

The bottom having insufficient grade to carry off the tailings with the limited amount of water at hand, a washing tank and sluice were put up on the side hill at an elevation of about 30 feet above the creek. The deposit is mined by a system of parallel trenches 12 feet wide, worked from the lower end of the deposit upward. Track is laid in these as they advance. The upper 6 to 18 inches of the over-lay are thrown off, the remaining $1\frac{1}{2}$ to 2 feet, together with the true grit (gravel) and 6 to 12 inches of the bed-rock, are shovelled into cars holding about half a cubic yard. They are trammed to the foot of the inclined plane (8), and are hoisted to the top of the washing-plant by a small friction-drum engine (3). This tank is built of plank and is about 50 feet long, 18 feet wide and 6 feet high. On one of the sides there is a door or opening 4 feet wide, reaching to within 4 inches of the bottom to a sill. The grit is dumped into the tank and a constant stream of water is kept flowing over it. The action of this stream is reinforced by water played on the material from a hose-nozzle under a head of 30 feet. This head is obtained from a stand-pipe (4) to which water is pumped from a reservoir (1) by means of a Hall duplex pump (2) with a 4-inch discharge. Excepting at the time of clean-up, the tank is kept nearly full of gravel, and under the combined action of the two streams of water, closely imitating natural agencies, a very good concentration of the coarser nuggets takes place in the tank. The material, partially assisted with a rake, flows over a grizzly (6), the bars of which are set $1\frac{1}{2}$ inches apart. The coarser pebbles and boulders are forked off, while the finer gravel and sand are carried down into a sluice (7) situated below the grizzly. The sluice is 400 feet long, 12 inches wide and 10 inches deep, and has an inclination of 6 inches in 10 feet. It contains only about 20 feet of riffles, and these are situated about 100 feet below the grizzly. Origi-

nally, the whole sluice was filled with riffles, but these were removed when it was recognized that they were superfluous for saving gold. At the present time the gold is seldom found below the first 4 or 5 feet of the riffles. The first hundred feet of the sluice aid in thoroughly washing and disintegrating the material before it reaches the riffles. The upper riffles consist of diagonal slots cut in 2-inch plank, which is laid in the bottom of the sluice. The lower riffles are of the longitudinal variety (see Fig. 6). The upper riffles, as well as the surface of the material in the tank, are examined every evening for larger

FIG. 5.



Rocker Used by Tributers.

FIG. 6.

Upper End. Lower End.

Riffles in Sluice-Box. Scale, $\frac{1}{4}$ inch = 1 foot.

nuggets. A complete clean-up is made at odd intervals, depending upon the richness of the material worked on, etc. The gravel in the tank is entirely worked down by means of the hose, the coarser nuggets are picked out by hand, and the heavy sand, together with similar material found in the bottom of the sluice, after taking up the riffles, are washed in a rocker. No quicksilver is used on the property, there being no fine gold whatever. A loss of gold would more likely be in the form of larger nuggets, which are overlooked in forking out the coarser material, or which, on account of their round form and size, would roll over the riffles to the tailing heap. One large nugget, of the shape and size of a hen's egg, was found on the latter. Clay balls (sluice-robbers) also cause considerable loss.

When working to full capacity, 25 men are employed at these mines—5 men work at the tank and sluice, 1 playing the hose and dumping cars, 1 raking gravel out of the tank, and 3 helping the material down the sluice and over the riffles, fork-

ing out the coarser pebbles. The latter force is necessitated by the limited supply of water and the desire to work as large quantities as possible. Their work might perhaps be assisted by the use of a much shorter sluice, and a somewhat steeper inclination of the same, without endangering loss in gold of such a coarse character. The remainder of the force, excepting foreman and engineer, are employed in digging gravel, taking up bed-rock, etc. An average day's output consists of 80 carloads, about 45 cubic yards of loose gravel. Two and one-half to three cords of wood are burnt a day, at 65 cents per cord. Labor is very cheap, 60 to 65 cents per day. These figures, with reasonable additions for superintendence, supplies, etc., will place the cost of mining gravel by this method at about 50 cents per loose cubic yard. From June until November, when the water-supply is very limited, the right of mining the gravel is let out to tributers, who turn in as royalty $\frac{1}{4}$ of the finer gold, including pieces up to 1 ounce in weight, and $\frac{1}{2}$ of the larger nuggets (above 1 ounce). The tributers work in pairs, one pitting and taking out the bed-rock while the other one manipulates the rocker (cradle), shown in Fig. 5. It is made up like a barrel, with half-inch staves, smoothed on the inside, and with solid heads, the latter being a little more than half a circle. One wheelbarrow-load is put in the rocker at a time. After the gravel is thoroughly disintegrated by a vigorous motion of the rocker, the pebbles, etc., are thrown out, and finally, by a light movement, the finer and heavier portions are examined closely by eye. It is practically a panning-process on a large scale. Fifteen minutes are occupied in cleaning up one charge.

The Mills Property, Burke County, N. C.

(Piedmont Mineral Company, Limited).

This property is situated near Brindletown, about 14 miles southwest from Morganton. It comprises an area of some 10,000 acres, including the eastern portion of Pilot Knob and the western flanks of the South mountains, being drained by the waters of Silver creek. The problem here presented is the reworking of old gravel-deposits by a simple hydraulicking process where the grade is sufficient, or, where this is not the

case, by raising the material to the fall-line by hydraulic elevators.

Geologically, it is located in the South Mountain belt. The general strike of the crystalline schists is N. 20° W. and the dip 20° N.E. The rocks are decomposed to considerable depth, reaching often 50 feet and at times 100 feet. The strike of the auriferous quartz-veins is N. 60° to 70° E. and the dip 70° to 80° N.W. These veins are usually from a knife-edge to several inches in thickness, and are too small to work individually. One vein from 12 to 18 inches in thickness has been explored but was found to be almost barren. The gravel deposits occupy the present stream beds and adjoining bottoms, and the ancient channels now covered with deep over-burden and extending into the hillsides which flank the mountain. From Pilot Knob and along its foot slopes, a number of these deep channels radiate in all directions.

The facilities for obtaining water for mining purposes are good, though beset with difficulties. The numerous streams which have their rise in the South mountains are small, though of unfailing flow throughout all seasons. It is practicable to collect their volume and lead it to the larger part of the mining ground in ditch- and flume-lines and reservoirs with sufficient head for sluicing and hydraulicking purposes. The chief impediment is in the loss of grade before the mining ground in the lower country is reached, owing to the deep and numerous indentations of the mountains which it is necessary to circumvent. It is impossible to water some portions of the side-hills except by pumping into reservoirs or by constructing expensive siphon-lines.

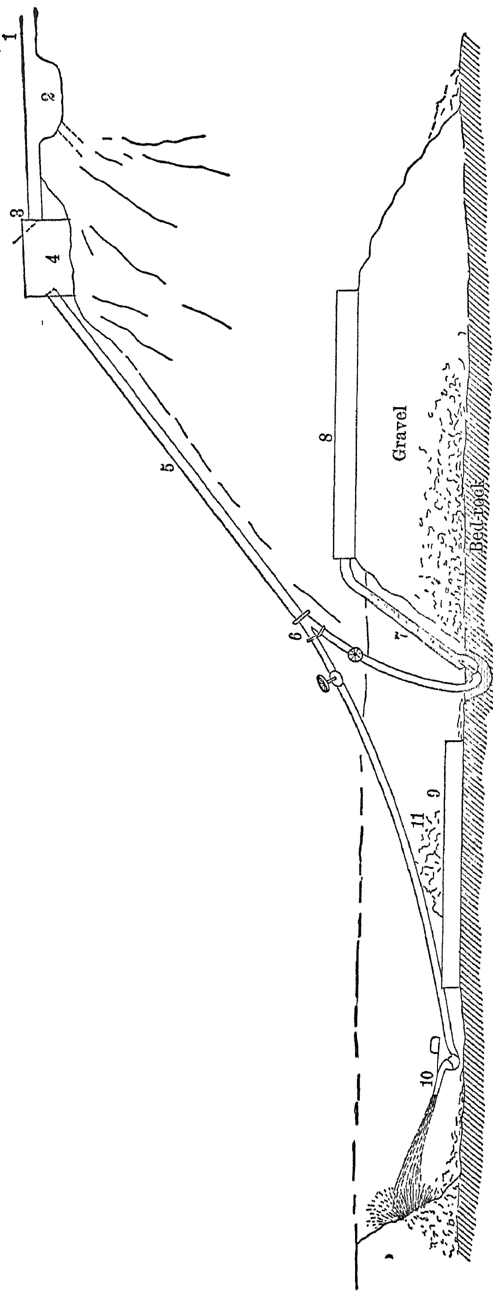
Brindle creek on the Mills property was the site of the first discovery of gold in this part of North Carolina in 1828. With few exceptions, most of the virgin placer-ground above alluded to has, by more or less continuous mining operations since then, been worked as high as water could be obtained with the present ditch lines. Much of the gravel has been washed over as many as three times. As no regular records have ever been kept, it is impossible to speak intelligently of the value of these gravel-deposits. Small channels yielding as high as $20 per cubic yard have been worked, but in general the ground will run nearer to from 4 to 50 cents. At present, the available

mining ground may be divided into two general classes: first, the bottom and ancient channel gravel-deposits; second, the decomposed country-rock in place, containing belts of small auriferous quartz-veins. Not much attention has been paid to the latter, excepting by tributers who in a spasmodic way have worked some deposits on the flanks of Brindle ridge, gouging out the small rich quartz-veins, and extracting the gold by crushing in hand-mortars and panning; they pay a royalty of $16\frac{2}{3}$ per cent. to the owner. Captain J. C. Mills at one time successfully worked one of these small quartz-belts by sluicing to a small stamp-mill (Dahlonega method), but the mill was destroyed by fire and never rebuilt.

In 1894 the Piedmont Mineral Company was formed with the object of again reworking the principal gravel deposits, obtaining as a by-product monazite, which occurs concentrated with the gold and is derived from the adjacent country-rocks by disintegration. At present, work on this property is concentrated at two points, the first in Silver creek bottom by giant and hydraulic elevator; and the second, by giant and continuous sluice-box on the Magazine or Parker branch. When the property was visited, July, 1895, the preparatory work was almost completed, and active mining operations were about to commence.

1. Silver creek forms one of the main drainages of the South mountains. The placer-deposits which are about to be reworked on the Mills property are situated near its head-waters. They are about 1 mile in length and are located mainly upon the west bank, on which the gravel often extends out a distance of 500 to 600 yards. The main difficulty encountered is the want of fall in the bed, a feature common to many Southern placers. It amounts in this case to less than 1 foot in 100. To overcome this obstacle for hydraulicking with continuous sluice, the use of the hydraulic gravel-elevator was decided upon, and some experiments were made with it a few years ago. In the main, they were satisfactory, but were soon abandoned, the plant being unfit for continuous use, and monazite not being at that time a valuable product. Fig. 7 gives a rough sketch of the plant and method to be carried out by the present company. Twelve miles of ditch and flume line (1) carry the water from a reservoir, through the Dan Sisk gap in the South

Fig. 7.

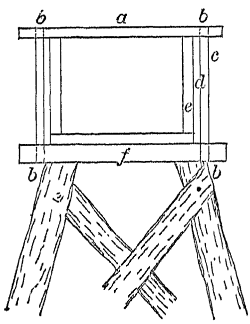


Sketch of Proposed Hydraulic Work on Mills Property, Burke County, N. C. (Not drawn to scale.) 1, ditch; 2, sand-pit with drain; 3, screen at head of penstock; 4, penstock; 5, pipe (10- and 8-inch); 6, Y branching into 2 7-inch pipes; 7, hydraulic elevator; 8, top sluice-boxes; 9, pit sluice-boxes; 10, giant; 11, bowlder and pebble heaps.

mountains, to a penstock (4), situated 200 feet above the level of the creek bed. The ditch is cut about 8 inches deep by 20 inches wide at a cost of about 25 cents per rod, and is given a grade of from $1\frac{1}{2}$ to 3 inches in 100 feet. The flumes are, at ordinary grade, 18 inches wide by 12 inches deep (see Fig. 8).

A sill, bent, top and side brace are erected every 6 feet at the joining-point and middle of each box. The bents are made of rough lagging seldom more than 6 inches in diameter, the greatest height of trestle being less than 30 feet. The sill of the flume acts as a cap for the posts. Wherever a small grade becomes necessary, the width of the flume is doubled. The cost of erecting these flumes is small, equal to about the cost

Fig. 8.



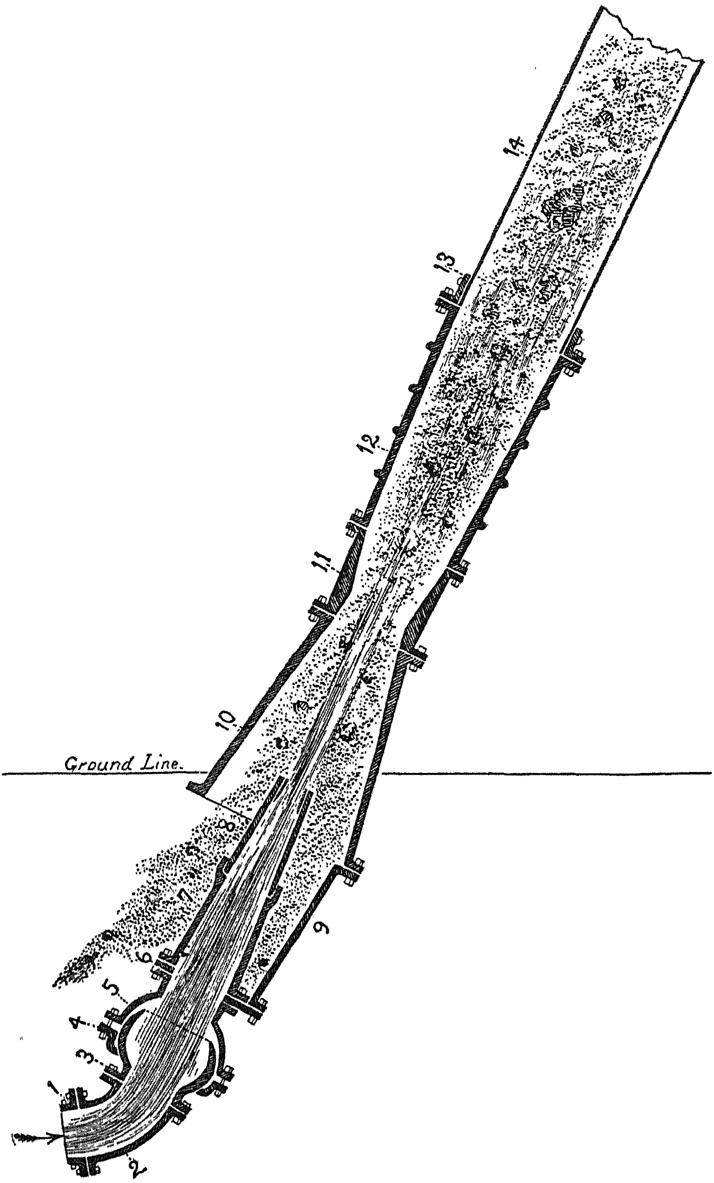
Flume, Mills Property, N. C. Scale, $\frac{1}{2}$ inch = 1 foot. *a*, 1 x 3-inch board; *b*, 1-inch holes; *c*, 1 x 3-inch board; *d*, wedging; *e*, $1\frac{1}{2}$ inch plank (sides and bottom); *f*, 2 x 4-inch sills and cap for bent.

of the material in them. Lumber is worth \$6 to \$7 per thousand.

The water, before reaching the penstock, flows through a sand-pit (2, Fig. 7), to catch sand, etc., washed into the ditch-line from the side. It then enters the penstock after passing through a screen (3) for removing leaves, sticks, etc. The pipe (5) running from penstock is 10-inch spiral riveted sheet-steel (with No. 16 Birmingham gauge), coated with coal-tar and connected with flanges. Smaller curves are made by placing cast-iron bevelled wings between the gaskets of the flanges, larger ones by suitable elbows. Near the gravel-pit the 10-inch pipe branches out through a Y (6) into 2 7-inch pipes, supplied each with a gate-valve, one leading to the giant (10) and the other to the hydraulic elevator (7). These are both of

California type and manufacture.* An illustration of the latter

FIG. 9.



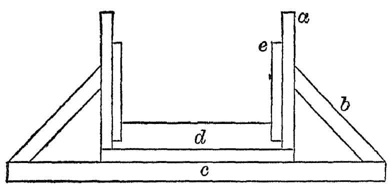
Hydraulic Gravel Elevator, Mills Property, N. C.

in detail is given in Fig. 9. The principle of this device is too

* Joshua Hendy Machine Works, San Francisco, Cal.

well-known to require a description. It is intended to keep the elevator stationary as long as possible, as its installation consumes considerable time. A pit must be sunk in the bed-rock, and as the elevator must also drain the workings (a drain on the top of bed-rock to the initial point of working was considered too expensive), the water would gain too much headway while the elevator is moved. The work in the main pit will be carried diagonally up the banks of the stream, so as to gain as much grade as possible. As soon as there is room, a sluice-box (9) will be placed between the working-bank and the elevator-pit. A cross-section of this is given in Fig. 10.

FIG. 10.



Section of Sluice-Box, Mills Property, N. C. Scale, $\frac{1}{4}$ inch = 1 foot. *a*, $1\frac{1}{2}$-inch surfaced pine plank (sides and bottom); *b*, 2 x 4-inch brace; *c*, 2 x 4-inch sill; *d*, 1 x 4-inch riffle; *e*, 1 x 8-inch sand-board.

The upper part of this sluice will be filled with 3-inch by 4-inch blocks and the remainder by 1-inch by 3-inch cross-riffles, placed 11 inches apart and held down by a sand-board, which is halved down on them. Both will help to protect the sluice-box against wear. All pebbles, etc., more than $\frac{1}{2}$ inch in diameter will be forked out of the sluices and left in the pit (11). After being raised by elevator, the material will pass through another sluice (8), the tailings from which will be worked for monazite. It is expected that by far the largest part of the gold will be saved in the first sluice.

2. The Magazine or Parker branch is a tributary of South Muddy creek. Its source is at the foot of Pilot Knob, and from the latter several gravel channels run towards it, sometimes entirely covered with soil, so as to make their location unrecognizable at the surface. One of these, the Magazine channel, has been extensively worked, first by open hydraulic work, and afterwards at the upper end, where the over-burden grew too heavy, by a tunnel, subsequently connected with the shaft. The former had a total length of 600 feet, and the latter

a depth of 50 feet. The creek-bed has also been worked, mainly with rockers. The present company propose to work this bottom, besides any side-hill channels they may find, by giant, sufficient fall being available to carry off the tailings in a continuous sluice-box below. Water for this work will be brought a distance of 5 miles to a large reservoir on the divide between South Muddy and Silver creeks, and from here in 2 miles of ditch and flume, along the foot of Pilot Knob, to a reservoir situated 100 feet above the creek bottom. This reservoir is designed to hold the water contained in the ditch after the gate at the large reservoir has been closed in the evening; and this will be the first water to be used in the morning, before that from the large reservoir has time to reach this point. The placer-deposit in the creek-bed has a total width of 400 feet. The old gravel-banks, etc., will be broken down and the material run into sluices similar to those described above, the tailings being carried down the branch to South Muddy creek.

The Chestatee Company, Lumpkin County, Ga.

The work pursued here, and its ultimate object, present special features of interest, and might warrant a greater application in the Southern gold-fields. The plant and property of this company are situated $2\frac{1}{2}$ miles from Dahlonega, Ga., on the Chestatee river, about $\frac{1}{2}$ mile above the entrance of Yahoolah creek. The property comprises about 250 acres of placer-ground on the banks of the river, together with about 1 mile of the stream-bed. The main object in view is to turn the river into a new channel, which is at present being excavated, and ultimately to work the stream-gravel as well as that in the adjacent bottoms. At the lower end of the property a dam has been thrown across the river and a substantial and well-constructed power-station has been erected, supplying power for a Blake duplex 12-inch by 24-inch pump and a 50 horse-power dynamo by 2 66-inch Leffel wheels. The latter were installed to furnish motor-power to a centrifugal sand-pump, used for raising gravel from the channel, but this has for the present been abandoned in favor of a hydraulic gravel-elevator. The substitution was made for economic reasons as well as for the fact that the latter had in its favor greater simplicity, more constant work and

easier portability, as well as greater ease of installation. The hydraulic elevator in use here is a recent design of Mr. W. R. Crandall, the general manager of the Chestatee Company. It combines cheapness and compactness of construction with good working results—a novel feature in the South being the introduction of air at the nozzle whenever the inlet of the elevator is entirely submerged. Water under 80 pounds pressure is supplied to this, as well as to the giant, by the Blake pump. This direct appliance of pressure without intermediate stand-pipe or reservoir has proved very successful, the only precaution necessary being to shut off the pump before closing the feed of the giant or elevator. It also has this advantage that, when occasion demands it, smaller nozzles can be used, and the pressure thus increased.

The channel is cut 30 to 35 feet wide, down to bed-rock in depth, and will have a total length of about half a mile. It runs almost parallel to the river, and from 50 to 200 yards from the north bank of the same. When completed, the waters of the river will be turned into it by means of a wing dam. The gravel above the bed-rock in this channel is auriferous, and is at present paying the expenses of the preliminary excavations. It averages 1 foot in thickness with 6 to 10 feet of over-lay. The latter is worked off during the night-shift (by use of electric light illumination), and the gravel thus exposed, as well as about 2 inches of bed-rock, is taken up during the succeeding day. The method of work and the management of the plant is similar to that proposed at the Mills property, N. C., and described above. The main difference lies in the fact that a long sluice-box is not used in the pit. A ground-sluice, naturally cut by the water from the giant running to the elevator pit, serves this purpose, and the soft slate bed-rock proves an admirable gold-saver. When work is far enough advanced, a small sluice-box 12 feet long is placed immediately preceding the elevator-pit, and there is always a similar one, filled with longitudinal riffles and round punched iron plates, on the bank at the head of the elevator pipe. But little gold is obtained from the latter.

The ultimate success of this work will be of great interest, and may stimulate other undertakings in this direction.

Chestatee River Dredge-Boats, Lumpkin County, Ga.

Dredge-boats of various descriptions have been at work on the Chestatee river for a number of years. The work has been spasmodic, and failures are more often recorded than successes. The river, where operated on, is about 100 feet in width and of variable depth. Numerous shoals make dredging difficult.

A steam-vacuum dredge* was operated for a time on this river; it did good work, especially in cleaning up the bed-rock. The main difficulty, and the reason for abandonment, was the banking up of the tailings around the boat, finally hemming it in. Dredging-operations using the principle of the hydraulic elevator† were also attempted, but proved unsuccessful. In July, 1895, there were two dredge-boats on the river, one above and the other below New Bridge. The former of these, operated by Mr. Frye, is on the principle of a continuous bucket-elevator. So far it has not been operated successfully, the buckets and continuous link-chain proving entirely too light for the work. The other boat has for some time been operated at a small profit by Mr. Jacquish. It was erected six years ago by the Bucyrus Steam Shovel Company, at an initial cost of about $15,000. After being worked for two years it lay idle until last summer.

The machinery is installed on a scow, 26 by 70 feet, drawing 3½ feet of water. It consists of a Bucyrus shovel (scoop) of 1¼ tons capacity, derrick and hoisting drums for operating the same, a small horizontal engine and centrifugal pump for supplying water to wash the gravel, and a 60-horse-power locomotive-boiler. A barge, 100 by 20 feet, lying alongside of the dredge-boat, carries the sluices. There are two lines of sluice-boxes, each 3 feet wide and 18 inches high, running the full length of the barge, and filled with longitudinal riffles, made up in five foot racks, composed of 1 by 3 inch slats set 1 inch apart. The gravel is discharged from the shovel on an iron shod platform at the head of these boxes, where the boulders and larger pebbles are removed. The gold is caught almost entirely

* For description, see *Gold*, A. G. Lock, 1882, p. 890.

† For a description of similar operations, see "A New Method of Dredging Applicable to Some Kinds of Mining Operations," R. W. Raymond, *Trans.*, viii., 254.

in the upper two racks; the tailings run off into the river in the back of the boat. When in favorable ground, the dredge will scoop and deliver an average of 1 bucket every two minutes. There are three men on the dredge-boat, engineer, fireman and craneman, and six men at the sluice-boxes. Work is carried forward up stream, the scow being moved against the current by anchoring the scoop and pulling the scow towards it by means of the crane-engine. The main wear and tear are on the lip of the scoop and on the chains. A steel lip 12 inches in length wears out in about six months. The river-ground is leased on a royalty of from 5 to 10 per cent. by the property-owners. It is said that gravel as low as 5 cents per cubic yard can be worked at a profit.

The Dahlonega Method, With Special Description of the Hedwig Mine.

The Dahlonega method of mining and milling is one which is particularly adapted to the large bodies of low-grade auriferous saprolite schists, such as exist in the Dahlonega district of Georgia. It consists in cutting down the soft decomposed ore-bodies by means of a hydraulic giant, the water from which carries the material through a line of sluices to the mill situated some distance below the workings, usually on the banks of a stream from which it derives its water-power. In the mill the coarser and heavier portions are retained by means of a screen, and are fed to the battery by hand, the mud and fine silt being carried through into the river. Generally, a third of the gold saved is caught in the riffles of the mine-sluices, the remainder being obtained in the mill.

Historical.—The Dahlonega method first originated in 1868 by sluicing the material from the mines to platforms near the mill, from where it was hauled to the mill in carts. This was improved by placing bins, with perforated bottoms, in the stamp-mills, from 4 to 5 feet above and in back of the mortars; underneath this bin was a settling-box, in which the sandy material settled and the slimes overflowed. At the Childs's mill, near Nacoochee, a plant was erected, consisting of a series of washing and sizing plate-screens, in which three sizes, coarse, medium and sand, were made and milled separately. It is stated that

Fig. 11.

Wrought-iron Siphon Pipe (3 ft. inside diameter), 2000 feet long, on the Hand Barlow Ditch Line, crossing the Yahoolah River, one mile from Dahlonega, Ga.

FIG. 12.

Dahlonega Method of Mining, showing Giant and Ground Sluice.

all the millable ore was saved in this way, in a clean shape, free from mud.

The present practice is to flush the material on to the mill-floor in back of the batteries, this space in the mill-house being practically arranged as a large bin with a slat screen (distance between slats about $\frac{1}{2}$ inch) at one end. Frequently a V-shaped storage-tank is situated outside of the mill, where the material is collected and flushed into the mill as occasion requires.

Water-Supply.—The system of reservoirs, ditches, etc., in this district is by far the most extensive and best equipped in the Southern gold-belt. The principal water-line is known as the Hand and Barlow ditch, having a total length of 34 miles, the main canal being 20 miles long, 6 feet wide and 3 feet deep, and furnishing 800 miner's inches. The grade averages 5 feet to the mile, being $4\frac{1}{2}$ feet on straight lengths, with slightly steeper grades on bends. The cost of digging this canal was about $1 per rod; the total cost, including trestling, etc. (excluding siphon-line), was $1000 per mile. The canal crosses the Yahoolah valley about 1 mile northeast of Dahlonega, in a wrought-iron siphon-pipe (see Fig. 11) 2000 feet in length. The difference in level of the two ends is about 6 feet. The inside diameter is 3 feet, the thickness of the pipe being $\frac{3}{16}$ of an inch in the upper and $\frac{3}{8}$ of an inch in the lower part. It was built in 1869.

Four miles from Dahlonega the water is carried across a similar depression in a wooden tube which is $\frac{7}{8}$ of a mile in length and 3 feet in outside diameter. It is made of 3 by 5-inch staves, trimmed so as to make a tight fit. These staves are laid in wrought-iron hoops, forming alternate joints; the last stave is driven in with a maul. This tube was built in 1868, and is still in good condition.

Auxiliary ditches run off from the main canal to the various mines. A portion of this water was formerly leased out at the rate of 12 cents per miner's inch for 24 hours. The present owners, The Hand & Barlow United Gold Mines and Hydraulic Works of Georgia, are, however, at present using the whole amount in working their own mines. Besides this system there are several smaller ones, bringing the total length of ditch-lines up to about 80 miles.

A unique feature of the water-supply at the Findley mine is the elevation of the water from the ditch-line to a reservoir situated 152 feet above it, by means of a hydraulic pumping-engine made by the Filer & Stowell Company, of Milwaukee, Wis. This pump is situated near the stamp-mill, 285 feet below the ditch-line. The water is led to it from the above ditch in a 16-inch straight-riveted feed-pipe 456 feet in length, and is discharged by it into a reservoir of 88,000 cubic feet capacity, a total vertical height of 437 feet, through a 12-inch steel pipe 1141 feet in length. The principle involved is that of the hydraulic ram, inasmuch as a large quantity of water under a lower head raises a certain portion of itself to a higher head, the remainder being waste. The machine, however, is of entirely different and, so far as known, novel construction. It is of the duplex pattern, the two engines being connected by gearing and with an 8-foot fly-wheel. Each engine has 3 cylinders in tandem, to which the water under the feed-head (123 pounds) is admitted and discharged by valves of the Riedler type. In one of these cylinders the water is raised to the greater head (190 pounds) at the expense of the feed-water, under head, going to waste in the other two. A shifting-valve is attached to the latter to give relief to the valves. The stroke is 18 inches, and at a high piston-speed of 250 feet per minute the pump works very smoothly. Tests had not been made, and no figures of efficiency could be obtained at the time of our visit. Such figures, as well as a more detailed description than could be made after a hasty examination, would be of great interest. The present working-capacity of the pump is 600 gallons per minute.

Mining.—The general character of the ore-bodies has been described on pages 673 to 678. The depth of the saprolites (decomposed schists) in the Dahlonega region reaches often to 50 and sometimes 100 feet. Enormous openings have been made in these by the hydraulic giant, whole sides of the mountain being torn off in places (see Fig. 12). The head employed in hydraulicking varies from 50 to 150 feet, dependent on the height from which water can be obtained. Where harder rock is torn loose, it is broken by hand-sledges and thrown into the ground-sluices. Powder is sometimes resorted to for breaking down the more resistant ledges. In order to shorten the dis-

tance in sluicing to the mill, tunnels are often run through the intervening hills (as at the Hand and Findley mines). The wooden sluice-line is supplied with longitudinal riffles throughout its entire extent.

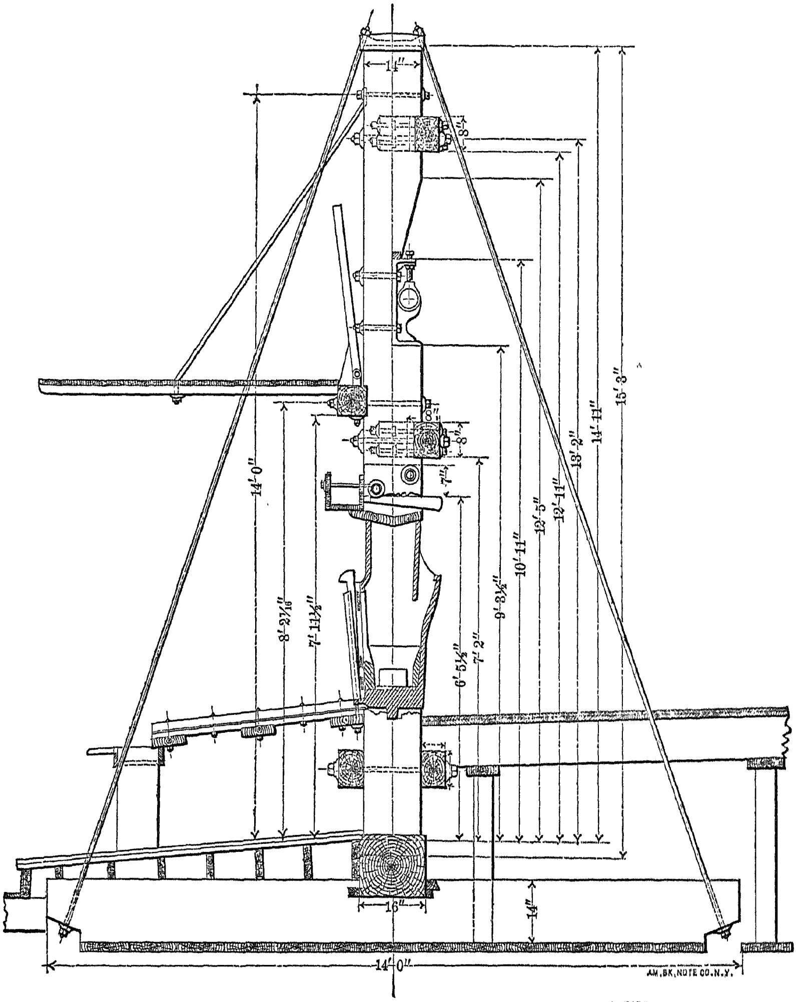
In the pursuance of this method a large proportion of the material carried to the mill is perfectly barren, for the reason that the entire mass is not gold-bearing, but only certain streaks of it, which cannot be mined separately by this method.

Milling.—The Dahlonega method of mining and the milling-material resulting from the same have developed a milling-practice particularly characteristic of this district. The material floated to the mill is of necessity of small size, the larger pieces of rock being sledged before entering the flume. Thus crushing is dispensed with. Automatic feeders at the mill have been tried, but were found impracticable, the variable hardness of the ore (only a small proportion being hard quartz and rock) making hand-feeding imperative.

The battery which is almost universally in use is that of the Hall type, invented and patented by Mr. Frank W. Hall, of Dahlonega. The usual weight of the stamp is 450 pounds. Figs. 13 and 14 give the two vertical sections of this mill. It represents novel features both in the battery and in the setting. The long battery blocks and a bed-rock foundation have been entirely dispensed with. The mill can be set upon any level piece of ground, a 2-inch plank platform forming practically the only foundation. The plan of construction (well shown in the drawing) makes the frame self-contained, the blow of the stamp and the reaction being absorbed and neutralized in the setting. Elasticity is maintained by the guy-rods. A suspended platform gives access to the props, cams, etc. The mortar is held in place by a rib on the bottom fitted in a corresponding gain in the mortar block. It is held down on the latter by wedges driven against blocks bolted on the inside of the battery posts. The small inside dimensions of the mortar are still more narrowed down by chilled-iron liners, which reach to within an inch of the dies. The main purpose of these liners is to bring the ore, on being fed, immediately under the shoes. They also protect the mortar against wear, and help to some extent in collecting and secreting amalgam. Quicksilver is fed to the batteries, and in some cases a considerable amount of amalgam

collected is obtained from the mortars. The liners are fitted with dove-tails and lugs at the end, and are finally held in place by two large keys driven against the screen frame, which is shod with wear iron on each side. On removing the front

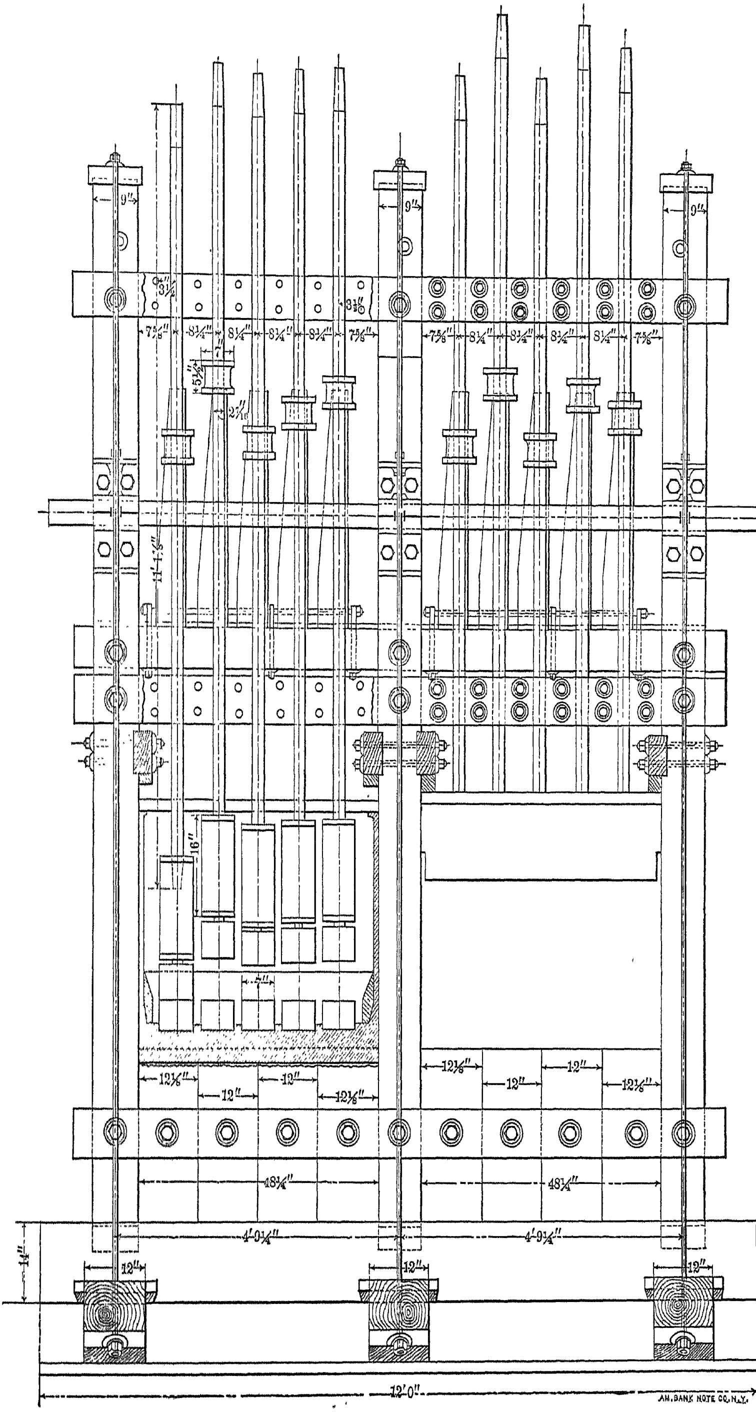
FIG. 13.



Vertical Cross-section of the 450-pound Hall Stamp-Mill.

liner the mortar is opened to the floor. The dies which sit in ½-inch depressions are easily withdrawn, the back and side liners drop out, and the mortar can be cleaned in a few minutes. The whole clean-up in a 10-stamp mill is accomplished in

Fig. 14.



Vertical Longitudinal Section of the 450-pound Hall Stamp-Mill.

the space of half an hour. The front liner determines the height of discharge, which, when the dies are new, is about 2 inches. An annealed copper plate, 4 feet long and of the full width of the mortar, is in most cases considered sufficient for the outside amalgamation. The weight of the 450-pound stamp-mill is divided as follows:

	Pounds.
Stem or spindle,	175
Head of boss,	150
Tappet with keys,	50
Shoe,	75
Total weight of stamp,	450
Die,	50
Mortar,	2100
Liners for same,	240

The average drop of stamp is 9 inches; number of drops per minute, 90. The whole machine is well constructed, and admirably fulfils its purpose of handling large quantities of the Dahlonega mill-stuff. The mill is also built with heavier stamps, and some slight changes are made in the frames of these. None of these heavier mills were seen in operation; but the setting employed is said to give as great satisfaction as in the lighter ones. Whether the application of this mill would be extensive for harder ores we are unable to judge. It certainly gives the extreme of rapid crushing, and might be adopted where such an object is in view.

The cost of these mills is light and that of installation small as compared with those of Western type.*

* The following figures were obtained in the camp as representing the average cost of a 450-pound 10-stamp mill of the Hall type, as erected and used in the Dahlonega district:

All iron-work for batteries and setting, including copper-plates (f. o. b. works, Cincinnati),	$700.00
Freight on same, and cost of erection, about,	500.00
Buildings, floors and sluices,	400.00
Engine and boiler, with connections,	600.00
Freight on same, about,	150.00
Total cost of complete mill,	$2350.00

Water-wheel and installation of same would cost about the same as engine and boiler. Chrome steel (made in Brooklyn, N. Y.) and Wilson pressed steel (made in St. Louis, Mo.) shoes and dies find about equal favor in the district, costing respectively 6 and 7 cents f. o. b., works. Cast-chilled iron shoes are also used to some extent, at a cost of about 3 cents per pound.

Mills similar to the Hall type are also made in Gainesville and Atlanta, Ga.

Almost all the mills in Dahlonega are operated by water-power, using turbines of the Leffel type for large quantities and low heads and wheels of the Pelton type when the water is small in quantity under a high head. The crushing capacity of these mills varies from 2 to 5 tons per stamp of 450 pounds in 24 hours, depending greatly on the nature of the material run through.

In hydraulicking, and subsequent transportation by water, a partial concentration takes place, resulting in the eventual deposition of a largely enriched product in the mill. The light stuff and most of the slimes pass through the mill, in almost all cases without subsequent treatment, and the heavy product remains, the enriching being all the way from 2 to 5 times the original value of the ore in place. Besides this, free gold (generally about one-third of the total amount saved) is caught in the sluices before reaching the mill. Some of the losses in this process are evident from the above. Another serious loss, which is rapidly making itself felt as the mines grow deeper and less decomposed ores occur, is that of gold in the sulphurets. In such ores that carry sulphurets at all it is stated that they will run from 2 to 10 per cent., the concentrates from which are reported to assay as high as $40 and higher. So far, concentration has not been carried out on a working basis. Despite many inquiries amongst local mill-men and others, we could hear no reports of losses in amalgamation resulting from so-called rusty gold. A loss of this nature was in a few cases ascribed to the finely-divided or flaky condition of the gold. Dr. Arthur Weld, vice-president and general manager of the Hand & Barlow United Gold Mines and Hydraulic Works of Georgia, has within the past year conducted numerous experiments relative to the inherent losses in the Dahlonega method. Among other experiments carried on by Dr. Weld may be mentioned that of chlorination in bulk.

It is difficult to give any average values of the Dahlonega ores, or in fact to clearly designate exactly what the term ore applies to in this district. Material worth as low as 40 cents per ton has been milled at a profit. If this figure per ton, plus the gold saved in the sluices (20 cents per ton milled) represents the milling-value of 5 tons of material mined, as is stated to be frequently the case, then the value of the latter per ton

must have been 12 cents. As a rule, however, the mill-stuff is of better grade than the above. The actual ore (quartz) is stated to assay from $1 up to exceptionally high values in the cases of rich stringers or pockets.

The cost of mining and milling throughout the district will average from 18 to 25 cents per ton of ore milled.

A description, somewhat more in detail, has been prepared of the following mine as representing perhaps most perfectly the Dahlonega method in its original type (of working soft saprolites or highly decomposed material).

The Hedwig Mine is situated near Auraria (formerly Nuckolsville) 6 miles west of Dahlonega. It consists of a large open cut about 60 feet in depth, run on a line of siliceous, micaceous ore-bearing schists, 60 feet in total width. The strike of the schistosity is N.E. and the dip 60° S.E. Three separate ledges of barren hornblende gneiss (brick-bat) enclose two ore-bodies, striking and dipping conformably to them. But very few small quartz-stringers occur in the mass. Water is furnished to the giant (3-inch nozzle) under a maximum head of 60 feet from a reservoir situated on the hillside above the mine. Six men are employed at the mine at 80 cents per day (day-shift only).

The material is run to the mill in a sluice line 2800 feet in length and 14 by 16 inches in cross-section, made of oak boards. It is supplied with longitudinal riffles made of 2 by 3-inch post-oak scantling. The grade of this sluice is 4⅞ inches in 12 feet at the lower, and 3½ inches at the upper end, that is, in the cut where it is not necessary to avoid overflows. The outside mill-bin holds about 240 tons, and the material is flushed from here to the inside bin, which holds 200 tons. Formerly there were three outside bins and the ore was hauled to the mill in cars.

The mill is a 40-stamp one of the Hall pattern, with a 12-foot driving-pulley. It is run by a Ridgeway wheel, using 40 inches of water from two 1-inch nozzles. The water is supplied from the same reservoir that furnishes the giant at the mine, by an 18-inch spiral riveted pipe-line, 2880 feet in length, under a head of 226 feet. The weight of the stamps is 450 pounds; drop 9 inches, 80 times per minute; discharge 2 inches; round punched screen, 121 holes to the square inch; length of plates (plain copper) 8 feet in two sections; ten of the stamps were fitted with silvered plates in 2-foot sections. Only the upper 4 feet

of the plates in the mill are kept in shape; it is stated that no gold was saved on the lower ones. The tailings flow off through mercury traps. The overflow from both the outside and inside bins runs through a short line of riffled sluice-boxes. Seven men are employed in the mill in two shifts, at 90 cents per day.

The Lockhart Mine, Lumpkin County, Ga.

The Lockhart mine is situated on the west bank of the Yahoolah river near Dahlonega, Ga. It represents the working of ore-bodies of the Dahlonega type by underground mining.

The Dahlonega method of mining the saprolites was formerly employed here, and the old open cuts, practically abandoned, are of considerable extent. This is the only mine in the Dahlonega district where underground work of any importance is being carried on.

The character of the ore-bodies at present being mined, consists of veins of the Dahlonega type (see descriptions pp. 676, 677), where the quartz-filling has been more extensive, in places occupying the greater part of the fractured gneiss bands, which in a mining sense may be termed the vein, the boundaries of the gneiss bands forming continuous smooth walls and being the limit of the minable ore. The normal strike of the schists at the Lockhart is N.E. and the dip S.E.; at one point, however, they bend around a mass of "brick-bat" schist, the strike being abruptly changed to the N.W. with N.E. dip.

The principal work is being done on the Blackmore vein, where the country is a biotite hornblende-gneiss. The strike of this vein is N.E. and the dip 30°–60° S. E. It varies in thickness from 3 to 6 feet. The ore-body is opened by two adit-levels on the vein, 60 feet apart. The lower one, which enters the hillside at a depth of about 135 feet below the original outcrop, has a length of 400 feet, and the ore has been stoped out between it and the upper level for a distance of 100 feet from the face, which is the length of the ore-shoot so far as explored. This shoot has also been worked from the upper level to the surface. The pitch is steeply to the N.E. The ores from this shoot mill from $4 to $5 per ton. Besides this richer shoot the bottom level exposes ore throughout its entire length. This,

however, decreases in quality as the mouth of the tunnel is approached, where it yields only $1. The system of work is underhand stoping, stulls being placed 6 feet apart to hold up the ground. The ore is carried from the stopes in barrows to a platform at the mouth of the tunnel, from where it is hauled to the mill by carts.

The same vein has also been opened by a shaft, 50 feet deep, at the mill house, which is situated about 300 feet N.E. from the mouth of the mine. A drift 300 feet long was run on the vein here, which is reported to be 14 feet thick, carrying highly sulphuretted ores, which milled $4. This part of the mine is now under water.

Other ore-bodies have been opened up to some extent, but not sufficiently to say much of their nature.

The ore is treated in a 20-stamp mill of the 450-pound Hall type, erected originally for working material from open cuts by the Dahlonega method. No crusher or mechanical feeder is used, and no concentration of the sulphurets has so far been attempted, although they are stated to be of high grade. The ore is fed by hand, one man attending to each ten stamps. The drop is 6 to 8 inches, 60 times per minute, and the discharge is about 2 inches high. The screen used is a No. 9 Russia slot. The plates are 6 feet in length, plain copper.

For the hard ores, such as are at present mined, this mill can scarcely be considered of the best type, being too light. A crusher and automatic feeder would also be applicable here, as well as concentrators and a subsequent treatment of the sulphurets. The mill-power is furnished by a turbine wheel, obtaining its head of water from a dam across the Yahoolah river.

The cost of production at the Lockhart is given as follows:

	Per ton of ore.
Mining,	$ .90
Hauling,	.15
Milling,	.20
Other expenses,	.10
Total cost of producing bullion,	$1.35

The average milling-value of the ore for the month ending February 3, 1895, is given as $4.15 per ton. No figures of the assay-values of the tailings could be obtained.

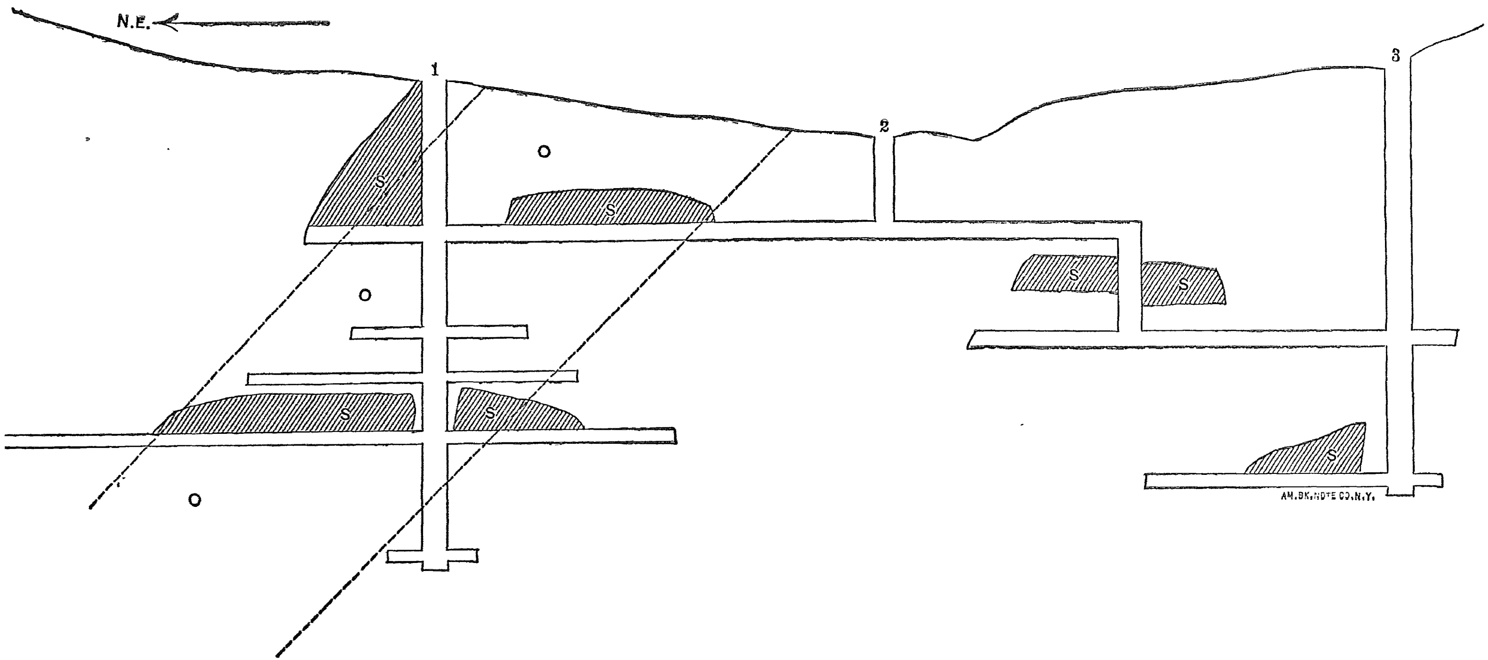
The Reimer Mine, Rowan County, N. C.

This mine is situated about 6 miles southeast of Salisbury on the waters of the Yadkin river. Geologically it is in the Carolina belt. It represents a highly sulphuretted quartz-fissure of marked persistency, with smooth walls and a clay gouge, the ore from which is worked by a stamp-mill amalgamation, concentration of the sulphurets, and their chlorination by the Thies process.

The vein is said to average 3½ feet in thickness, varying from 1½ to as high as 9 feet. The strike of the outcrop, which has been traced for 2 miles, is in an east and west direction. The dip is practically vertical. The sulphurets, mostly pyrite with a little chalcopyrite, occur in bunches, averaging about 10 per cent. of the ore. The quartz is compact, white and glassy. The wall-rock is a coarse crystalline eruptive, probably a quartz-diorite, and also a fine-grained phase of the same.

Until 1884, when it was destroyed by fire, a concentration-plant was in operation here. The concentrates which were obtained without previous amalgamation, were treated at the Yadkin Chlorination Works near Salisbury. Work was not taken up again until 1894. Fig. 15 gives a vertical section of the mine along the strike of the vein. Work is at present concentrated at the bottom of No. 1 shaft (1), at a depth of 190 feet. The shaft is poorly constructed and very wet. A Cornish pump, driven by a belt from the crank of a small friction-clutch hoisting engine, raises the water from the bottom into a crude ring at the 150-foot level, from where a No. 9 Cameron sinking pump raises it to the surface. No development is carried ahead, the ore being taken out by overhead stoping as soon as found. It was stated by the management that the poor condition of the mine and the crude method pursued, was due to the more or less experimental nature of the present underground developments. The size and substantial construction of the mill and chlorination plant seem, however, to have gone beyond this stage. On account of the limited development the mine is worked in three shifts of eight hours each, with two miners and helpers on each shift, who are paid respectively $1.50 and $1. Engineer, fireman and top-labor work in two shifts of twelve hours each. No definite information could be gained regarding cost of mining; but under the conditions existing, it must be ex-

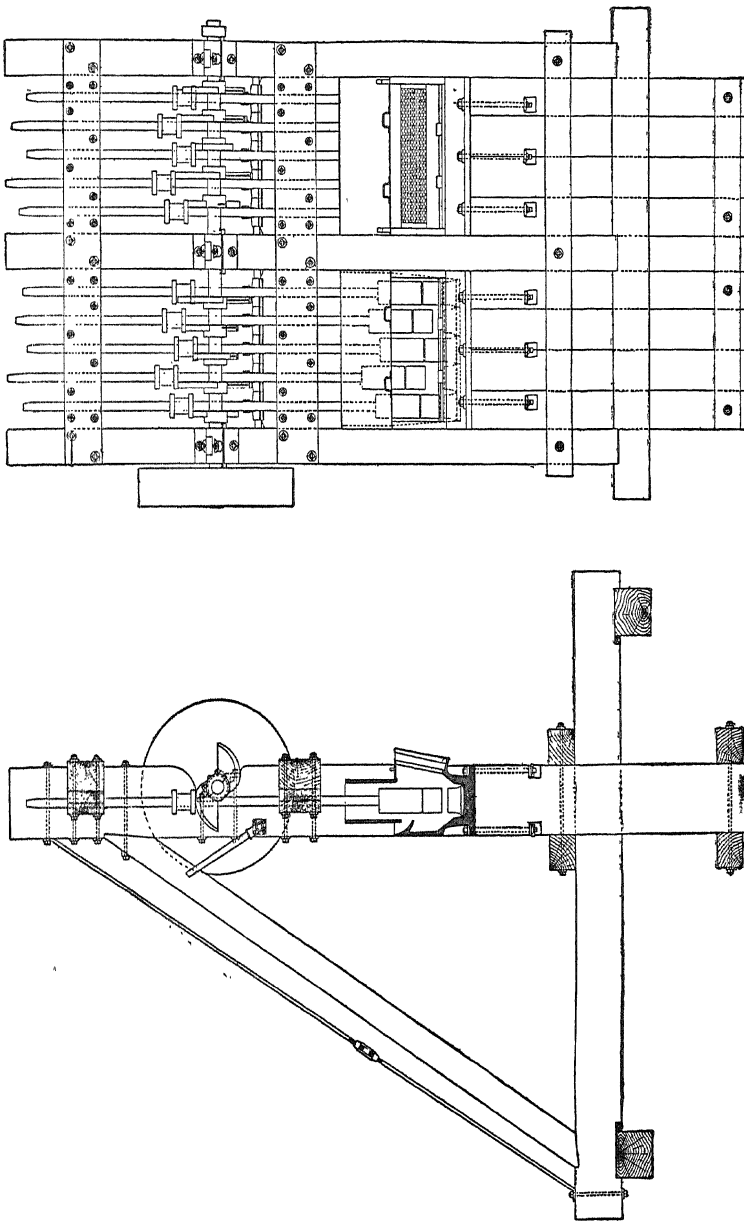
Fig. 15.



Vertical Section on the Strike of the Vein, Reimer Mine, N. C. Scale, 1 inch = 80 feet. 1, shaft No. 1; 2, air-shaft; 3, shaft No. 2 O, dip of ore-shoots; S, stoped ground.

cessive. The mill is a 20-stamp one, built by the Mecklenburg Iron Works.* The mortar (Fig. 16) is of a modified Cali-

FIG. 16.



Mecklenburg Iron Works, 750-pound battery, Reimer Mine, N. C.

* The Mecklenburg Iron Works of Charlotte, N. C., Captain John Wilkes, Manager, make a speciality of gold-mining and milling machinery. They have

fornia type, and of medium width and depth. A novel feature in this mortar is a large opening above and in back of the screen by which the inside of the screen can be reached to clear it of foreign clogging matter. The inside plates may also be taken out through it without disturbing the screen. The weight of the stamp is 750 pounds, given 5- to 7-inch drop, 90 drops per minute. No inside plates are used at this mill. The height of the discharge is 5 inches when the dies are new. The screens are 40-mesh brass wire. The outside plates are similar to those at the Haile mine (see p. 779). Only 10 stamps are at present geared up ready for work, milling 9 tons in 12 hours. About $\frac{1}{5}$ of the gold extracted is saved by amalgamation. The tailings from the plates are concentrated on 2 Frue and 2 Triumph vanners, producing about 1 ton of concentrates in 12 hours, running from \$30 to \$40 per ton. The two machines working side by side are stated to give equal satisfaction.

The concentrates are roasted in a large reverberatory furnace located in the mill-building, the hearth of which is $9 \times 41\frac{1}{2}$ feet. The capacity of this furnace was stated to be 4 roasted tons in 24 hours at a cost of \$1.25 per ton. The furnace is worked in two 12-hour shifts with two men on each shift, head roaster at \$1 and helper at 85 cents. Two cords of wood at \$1.25 per cord are burnt in 24 hours.

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lately erected a 5-stamp test-mill at their works, connected with a complete chlorination test-plant with a capacity of half a ton of raw concentrates per day. As being of interest and value in a paper of this kind, we have obtained from them the following list of the cost of milling- and chlorination-plants, erected in the South. The figures given are outside ones, and apply in each case to a complete automatic plant.

The cost of the machinery for a 10–750-pound stamp-mill with grizzly, crusher, self feeders, silvered inside and outside plates, Triumph concentrators (4 to 10 stamps), engine and boiler, together with all attachments, and plans for erecting and locating machinery, is given at \$5700 f. o. b., Charlotte, N. C. The same for a 20-stamp mill is \$10,350.

The complete cost of a 10-stamp mill as above erected (in the vicinity), will be about \$8000. Of a 20-stamp mill, about \$14,000.

The approximate cost of a 1-barrel chlorination-plant with two reverberatory furnaces, erected, is given at \$5500. The same for a 2-barrel plant with four furnaces at \$9700.

The complete cost of a 10-stamp mill with concentrators, roasting-furnaces and a Thies chlorination-plant with all necessary power and expenses may be figured at \$1200 per stamp. For a 20-stamp mill at \$1000 per stamp and for a 40-stamp mill at \$900 per stamp.

The price of shoes and dies of a chilled charcoal iron mixture is 3 cents a pound f. o. b. works.

The chlorination is carried on in a 1-barrel plant with a capacity of 4 tons of roasted concentrates per 24 hours. The building is arranged for the addition of another barrel which would allow the same work to be done in 12 hours, giving better opportunity for precipitation, and reducing the total cost of chlorination. The charge and the method of working is identical with that pursued at the Haile mine (see p. 783). No satisfactory figures regarding the value of the tailings from either the concentration or chlorination could be obtained: the figures given were high as compared with those of other mines. The cost of milling, concentration, roasting and chlorination per ton of ore milled is given at $1.80 per ton. This excessive cost, almost three times as much as that at the Franklin mine, an almost identical case as far as the plant and the size of the ore-body are concerned, must no doubt be largely laid to the fact that an attempt is made to supply a plant with a nominal capacity of 40 tons in 24 hours from a mine, in which the development does not warrant an output of 10 tons in this time.

The percentage and value of concentrates given above, with the addition of the gold saved on the plates, gives an estimated value of from $4 to $5 per ton to the ore mined, without including in this value the gold lost in tailings. Such an ore if found in sufficiently large bodies on developing the mine, should pay a profit with the present method of treatment under a close management.

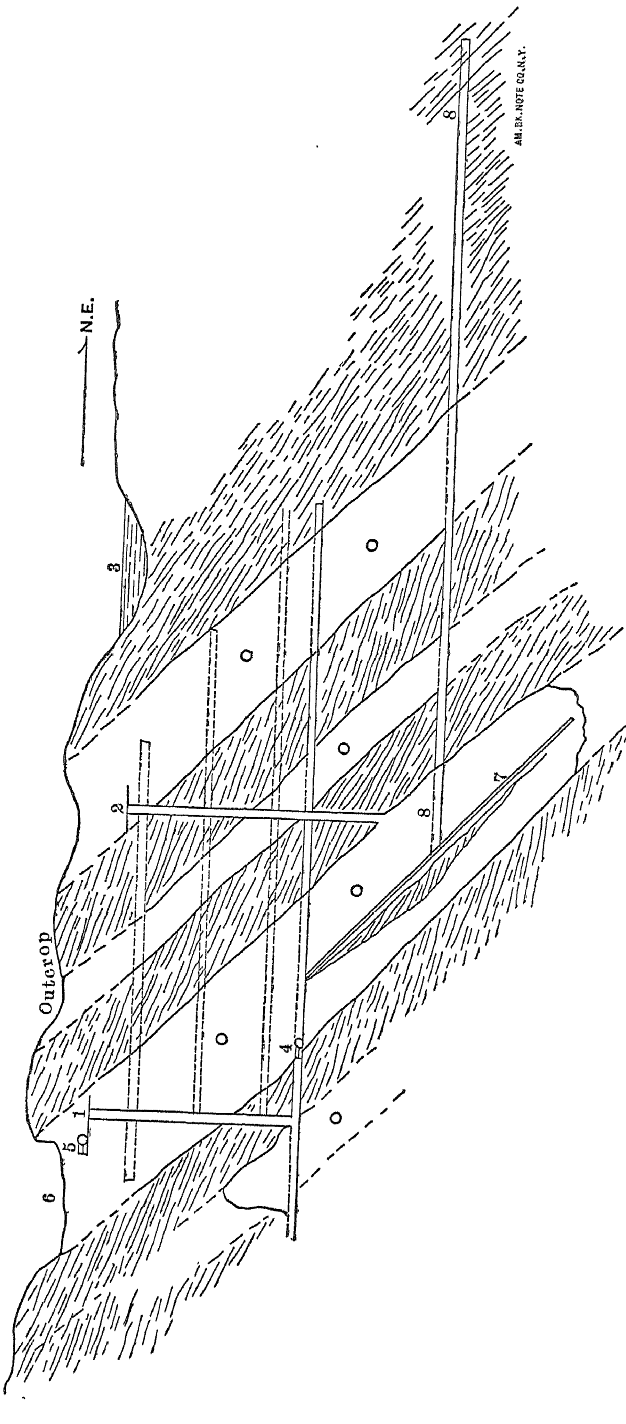
At the present writing, the mine is again closed down.

The Franklin Mine (Creighton Mining and Milling Company), Cherokee County, Ga.

This mine is situated on the Etowah river, about 16 miles northeast of Canton, the county-seat. Geologically it is in the Georgia belt. The proposition presented here is in most respects similar to that at the Reimer mine.

The country-rock consists of gneissoid mica and hornblende schists, often garnetiferous. The average strike is N. 55° E. and the dip 40° S.E. Granite dikes are stated to exist in the vicinity of the mine, but none have been, as yet, found intersecting the ore-bodies. The character of these ore-bodies has already been described. There are two parallel veins about

FIG. 17.



Vertical Section along the Strike, Franklin Mine, Ga. Scale, 1 inch = 200 feet. O, Ore-bodies; 1, No. 2 shaft; 2, No. 1 shaft; 3, Etowah river; 4, Underground hoisting engine; 5, Surface hoisting engine; 6, Franklin Pit; 7, Slope; 8, 350-foot level.

150 feet apart, known respectively as the Franklin and the MacDonald. Of these, the Franklin has been most extensively opened, and is the only one being worked at present. The strike and dip of the veins are, in the main, coincident with those of the country schists. The minable ore exists in lenticular shoots or cylinders pitching 45° N.E. (see Fig. 17). Four such shoots have so far been opened in the mine within a horizontal distance of about 750 feet on the strike. The largest one of these has a maximum length of 120 and maximum width of 14 feet. The average thickness of the ore-bodies is probably about 3 feet. All but one of the ore-shoots crop out at the surface, and they show considerable permanency in depth. The 350-foot drift in the mine has been extended in a northeasterly direction about 400 feet beyond the last ore-shoot. Although a permanent vein with clay casings, and in places heavy quartz-filling, has been found, the ore was not rich enough to mill. It is proposed to extend this development-work still further, with the hope of striking another ore-shoot. On the 235-foot level a horizontal diamond-drill hole (over 150 feet in length) was bored in the hanging, but no other parallel ore-body was found. Cross-fissures, from 3 to 6 inches in thickness, are met with in the mine, striking N. 30° to 35° W., with a vertical dip, and intersecting, though not faulting, the ore-bodies. These fissures are filled with coarse crystalline calcite, sometimes carrying inconsiderable amounts of pyrite. The structure of the vein-quartz at the Franklin is banded, and its character is milky, glassy. The sulphurets consist mainly of coarse crystalline pyrite (with very little chalcopyrite), usually occurring in bunches. Although the ore is over 50 per cent. free milling, gold visible to the eye is of very rare occurrence. The fineness of the gold is 980 to 989.

The property of the Creighton Mining and Milling Company comprises some 1800 acres. The first work done here was by open cuts in the outcrop of the ore-shoots. After the death of Mr. Franklin, the original owner, the mine was worked for a long time by his widow.

Before the adoption of the chlorination process for treatment of sulphurets by the present company, a cyanide-plant was erected and operated for a short time (see pp. 685 and 686).

The present condition of the mine is shown in Fig. 17,

giving a vertical section along the strike. The mine is worked entirely through No. 2 shaft (1) driven in the hanging-wall to a depth of 215 feet, at which point it strikes the vein. From this level work is carried on to a total depth of 430 feet by a slope on the dip of the vein and the pitch of the ore-shoot, resting on a small horse of poor ore.

The method of mining the ore is as follows: Levels are run every 100 feet, and the ore-lenses are entirely stoped out, leaving the intervening bodies of low-grade material as pillars. The levels are connected by a series of raises, their number depending upon the length of the ore-shoots. The ore is then stoped by underhand work, the raises acting as ore-chutes (mill-holes), and the cars being loaded directly from pockets in the level below. No pillars are left below the levels, the track, when necessary, being carried over the worked-out stopes on stulls. Only such timbers as are necessary to assist the men in their work are used, the walls requiring no support. All the material stoped is hoisted and milled, leaving no waste filling in the mine. Air-drills are used almost exclusively; for stoping, a Baby Rand with $\frac{7}{8}$-inch steel is used, while drifting is done with $3\frac{1}{4}$-inch cylinder Sergeant machines. The ore is raised in cars of $\frac{1}{2}$-ton capacity, first up the incline by underground hoisting engine (4), and then trammed to the bottom of the vertical shaft, from where they are hoisted to the surface on cages. No. 1 shaft (2) is used for ventilation and as a pipe-way. The mine is not a wet one, a small steam-pump, situated immediately below No. 2 shaft, taking care of the water. At the surface, the ore is run over a grizzly and then through a crusher, the jaws of which are set $1\frac{1}{2}$ inches apart. The crushed ore is hauled to the mill by mules in cars of $1\frac{1}{2}$ tons capacity, which are loaded from a bin below the crusher.

Besides the above deeper developments, exploratory work is being pushed at two other points along the outcrop of the vein, known as No. 3 and No. 4 shafts, and located respectively $\frac{1}{4}$ and $\frac{3}{4}$ of a mile to the southwest of No. 2 shaft. At both points inclined shafts are being sunk on the dip of the vein with the object of developing in depth lenses which have been worked to some extent on the surface.* Considerable diamond-drill-

* These two shafts are at present being pushed, and the milling-capacity will shortly be increased, September 8, 1895.

ing has been done on the property (some 800 feet in all) at a cost of about $1.25 per foot.

The mill is situated about ¼ of a mile from No. 2 shaft, on the east bank of the Etowah river. Water at a head of 7½ feet is supplied, by a dam thrown across the river, to two turbine wheels. One of these, a 60-inch Leffel wheel, supplies 23 horse-power to the stamp-mill, while the other, a 56-inch Davis wheel, drives a duplex Rand air-compressor. The concentrators are run by steam-power, that derived from the turbine not being of sufficient regularity to secure a uniform product. There are 20 stamps in the mill, 10 of Western make and 10 erected by the Mecklenburg Iron Works. Weight of stamps 850 pounds, 7-inch drop, 70 drops per minute, 6-inch discharge. No inside plates are used and no quicksilver is fed to the battery (a little coarse gold is cleaned from the battery sands). The screens are No. 7 slotted Russia iron, corresponding to about 30-mesh. The outside plates have the full width of the mortar. They are 8 feet long, arranged in four steps, and are handled in the same manner as those at the Haile mine. About 55 per cent. of the gold extracted from the ore is saved by amalgamation. The ore is fed from bins by Hendy automatic feeders. The mill handles 35 tons in twenty-four hours.

The pulp from each 10 stamps is carried by launders to four hydraulic classifiers, the overflow from all of these going to one slime-spitzkasten of 9 by 9 feet surface dimensions. The product of the 8 hydraulic classifers goes to 8 Embrey tables, the product of the slime-kasten being distributed to 2, making 10 tables in all working on mill-pulp. Besides these, there are 3 tables working on old amalgamation-tailings, assaying about $3 per ton. The concentrates are not clean, containing about 50 per cent. of sand, but close work would decrease the percentage of extraction. The average amount of sulphurets in the ore mined is about 5 per cent., sometimes running as high as 9 per cent. As high as 5¼ tons of raw concentrates are produced and treated in twenty-four hours. The tailings from concentration run at present about 85 cents per ton, giving a remarkably high percentage of extraction.

The concentrates are roasted in two double-hearth reverberatory furnaces, with a capacity of 2 tons of roasted ore each in 24 hours. Twelve pounds of salt per ton are added to the charge to change the carbonate of lime present to chloride.

Chlorination of the roasted concentrates is carried on in a one-barrel chlorination-plant, the arrangement of the same, and the method pursued, being identical with that at the Haile mine. The tailings from the chlorination run about 60 cents per ton, giving an extraction of over 95 per cent.

Labor, Costs, etc.—About 90 men are on the pay-roll of the Company when work is going on at full capacity. The force of men is variable, depending upon the output, and the amount of development work. The wages are as follows:

Drill runners,	$1 55
Helpers,	1 00
Muckers,	75 to 80 cents.
Trammers,	1 00
Blacksmiths,	2 25
Carpenters,	2 50

Three men are employed on each 12-hour shift in the mill and concentration-house, at the following wages:

	Per day.
Amalgamator,	$1 40
Concentrator,	1 35
Helper,	75

Roasting.—Two men on each shift, at $1.25. Cost of roasting, per ton of roasted concentrates, $2.00.

Chlorination.—One man on each shift. Cost of chloridizing, per ton of roasted concentrates, $1.48.

Supplies.—Timber, $9 per 1000 feet. Cord-wood, $1.25 per cord, 8 cords used per day.

Cost per ton of ore mined:

Mining, crushing, and tramming to mill,	$2 05*
Milling, roasting, and chlorination,	65
Total,	$2 70

The Brewer Mine, Chesterfield County, S. C.

The Brewer mine (the De Soto Mining Company) is situated on Lynch's creek, about 13 miles by road northeast of Kershaw, the nearest railroad station; it is about 8 miles (air-line) northeast of the Haile mine.

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* This figure includes all development work. The average value of the ore and the concentrates cannot be given for private business reasons.

The mining problem presented here is the working of large bodies of low-grade, sulphuretted ores by quarrying, milling, concentration, and chlorination.

Geologically, the mine is situated in the Carolina belt. The country-rock is a hard, devitrified acid volcanic (probably quartz porphyry), of a light bluish-gray color, resembling hornstone or chert. It is, in part, sheared into silicified, sericitic schists, similar to the slates at the Haile mine, though more highly silicified. Masses of coarse, pyroclastic breccia were found in the bottom of the large mine-pit, but the rock was not observed in place. The strike of the siliceous schists is very much confused, being in all directions; the normal strike is, probably, something like N. 70° E., and the dip 60° N.W. Numerous coarse-grained granitic dikes (G, Fig. 18) intersect the country, and the local abnormal strikes and dips of the schists may be due to their intrusion. These rocks occupy an elevation known as Brewer hill, which rises some 200 feet above the level of the main drainage, Lynch's creek on the east and Flat creek on the west. A heavy diabase dike lies on the west bank of Flat creek, and to the west of that the country-rock is granite.

The ore-bodies at the Brewer are similar to those of the Haile mine, being auriferous pyritic impregnations in the country-rock, and assuming more or less lenticular forms. Free gold appears as thin films or coatings on the cleavage and joint-planes of the schists. The ore-bearing rock is decomposed, in certain streaks more than in others, to the deepest workings of the mine, 150 feet, resulting in soft, friable masses which disintegrate into finely divided white sand. Certain portions of the deposit are richer in gold, and these also have an imperfect lenticular shape, from 10 to 30 feet in thickness. (O, Fig. 18.) These better-grade ores will run from $5 to $7 per ton, assay value, while the average run of the mine is in the vicinity of $3. The fineness of the gold is from 970 to 984. The total width of the ore-bearing ground is stated to be 800 yards. The body at present being worked has been opened in ore for a distance of 600 feet in a north and south, and 250 feet in an east and west direction. The sulphurets contained in the ore are finely divided pyrite, the percentage of which averages about 7 per cent. In one portion of the mine enar-

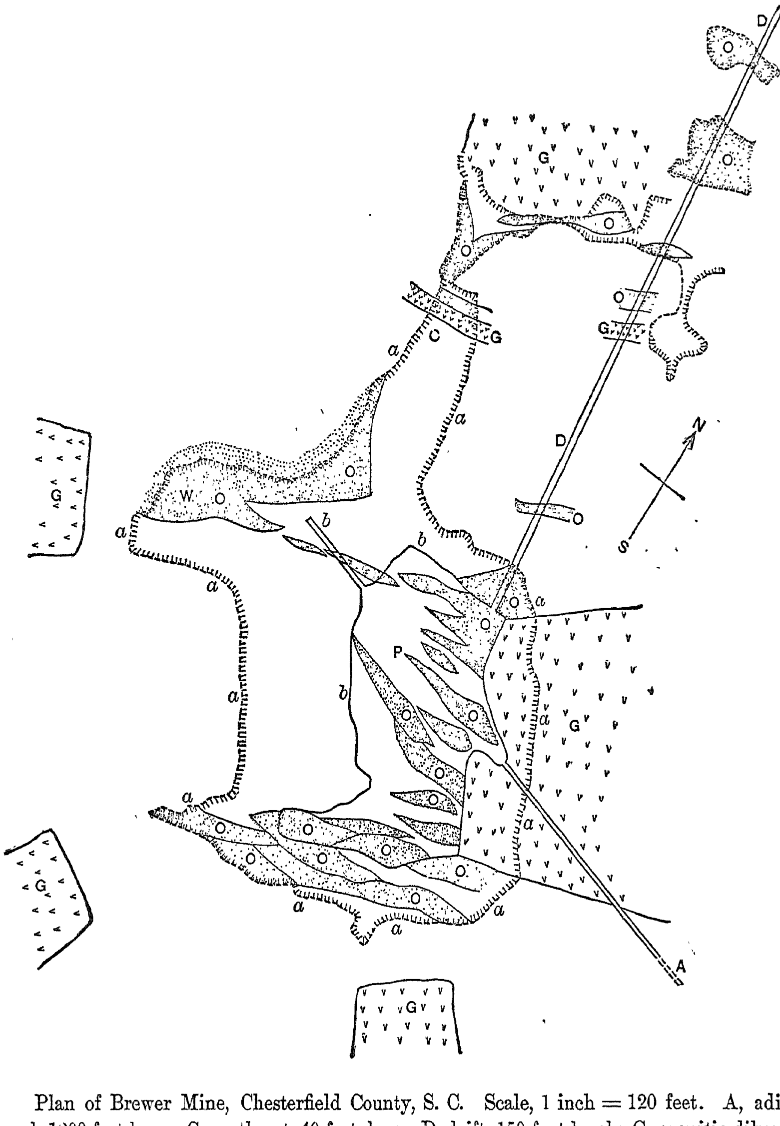
gite (and perhaps also covellite) appears in some quantity, but its occurrence is local. Other sulphurets occur in small quantities, but are interesting merely from a mineralogical standpoint. Tin-stone (sometimes in direct association with gold) has been found in hydraulicking at the Tan-Yard deposit; and pyrophyllite occurs as an alteration-product in the granitic dikes.

The ore itself is practically devoid of auriferous vein-quartz; small recticulated fissures filled with barren quartz intersect the country; and in the Tan-Yard, an old gravel-channel to the east of the mine, a large barren quartz-vein, 5 to 20 feet in thickness, is seen.

The Brewer mine, probably one of the first in South Carolina, was opened in 1828 by shallow pits in the saprolites, and in the gravels of the Tan-Yard, the material being worked in rockers. This work continued until 1857, and it is stated that in various years during this period as many as 100 to 200 hands were employed at one time, making from $1.50 to $3 per day, and paying nearly 30 per cent. royalty. From 1857 to 1862 Commodore Stockton mined and milled the ore in arrastras and Chilean mills. Up to 1879, when the Brewer Mining Company took hold of the property, there seems to have been a lull in the activity of the operations. In this and succeeding years the old Tan-Yard placer was reworked by hydraulicking. This deposit is an old river-channel, and was extensively worked in former days, being, in fact, the site of the first discovery of gold on the property. The width of the channel is from 200 to 300 feet, and its length about 1½ miles; it is now intersected by a large valley. The original overlay was about 6 feet, and the gravel from 3 to 6 feet in thickness, underlain by a thin bed of compact conglomerate, cemented by iron oxide; the bed-rock is a siliceous sericitic schist. The old miners in working this deposit did not wash the overlay, nor did they take up any part of the bed-rock. In reworking, the whole mass (from 5 to 20 feet in thickness) was hydraulicked, and as much as 4 to 5 feet of the loose bed-rock was also torn up. Water was pumped about 200 feet in vertical height, from Lynch's creek to a small reservoir situated at the head of the placer, from where a portion of it was delivered to the giant (2½-inch nozzle), by a force-pump under a pressure of 80

pounds, and the remainder run directly through the ground-sluices to carry off the tailings. Six men were employed in

FIG. 18.



Plan of Brewer Mine, Chesterfield County, S. C. Scale, 1 inch = 120 feet. A, adit level, 1200 feet long; C, north cut, 40 feet deep; D, drift, 150-foot level; G, granitic dikes; O, streaks of best ore; P, bottom of main pit, 150 feet deep; W, west cut, 50 feet deep; *a a*, surface line of open cut; *b b*, 150-foot level.

cleaning bed-rock, and two at the sluices. It is stated that a handsome profit was realized by this work.

In 1885 a 5-stamp mill was erected and run on ores produced in prospecting-work. In 1887 an adit-level (A, Fig. 18), 1200 feet in length, was driven into the hillside under the main ore-deposit, and the mine was opened from below by a raise, which was at the same time used as a chute, connecting with the open pit above. The stoping was carried on overground, and the material taken out through the tunnel. In 1888 a 40-stamp mill was erected, and started up in May, 1889. A Thies chlorination-plant was added in 1892, and operated for a short time during 1893. Until June, 1895, the mine was idle, and at that time preparations were being made for starting work.

Fig. 18 represents the plan of the Brewer mine as it is at present developed. It consists of the large open pit (P), 150 feet in depth, about 200 by 250 feet on the surface, and 100 by 180 feet in the bottom. The ore-body has been further explored by a drift (D), on the bottom level, extending 430 feet in a northerly direction, and being in ore all the way. The tunnel (A), is laid with narrow-gauge track, over which the ore is hauled to the mill by a small locomotive. This tunnel is drained by a wooden gutter situated in the center of the track line. At present ore is being quarried in the west cut (W), near the surface, from where it falls to the bottom of the pit (P), and is hauled to the mill through (A). The 40-stamp mill, which was not in operation when visited, is situated about a quarter of a mile east of the mine, on the west bank of Lynch's creek. It is of the Western type, built by Fraser & Chalmers. The weight of the stamps is 900 pounds. The mortars are 15 inches wide at the lip, and are fitted with front inside plates and 30-mesh steel wire screens. The outside plates of silvered copper are 8 feet long by 54 inches wide. Below the plates is situated a line of pointed boxes, serving simply as amalgam-traps, which discharge 2 feet above the bottom to four Frue vanners with 6- by 14-foot belts. This is one of the most substantial and best constructed mills in the South.

The chlorination-plant consists of 2 revolving pan-furnaces, 2 barrels, 8 filters, 2 stock-tanks, and 8 precipitating-vats of the same construction and arrangement as at the Haile mine.

When the mill was last operated (in 1893), the object was to put through as much material as possible; 5 to 6 tons of ore per stamp were milled in 24 hours, with 4-inch drop, 90 drops per

minute, crushing through a 20-mesh screen. Naturally, the pulp flowed over the plates without a large portion of it coming in contact with them; and, with only 4 vanners, the ultimate loss in tailings was so great as to leave little if any profit. The concentrates that were obtained ran from $15 to $20. About 50 per cent. of the gold in the ores is free, and of the amount saved in amalgamation 50 per cent. was in the battery and on the inside plate. The cost of mining and milling at the Brewer mine, as practiced above, is given at 75 cents; and the total cost (including maintenance, salaries, etc.), at $1 per ton of ore mined.

At the time of our visit it was proposed to start the mill in the near future on the soft decomposed ores, crushing about 2½ tons to the stamp, with an expected saving of about $1.25 by amalgamation.

Laboratory experiments with cyanide, and others with chlorination in bulk (the latter by Mr. P. G. Lidner), have been tried at the Brewer, but proved unsuccessful.

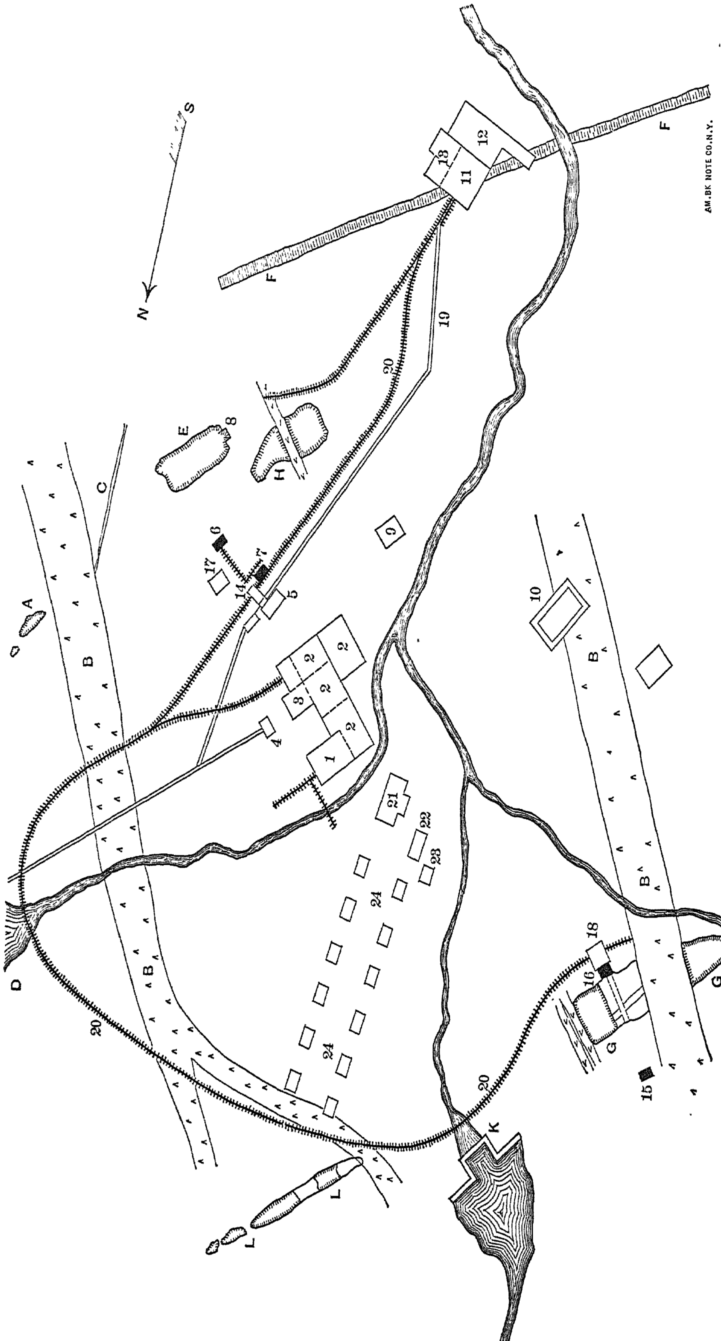
The Haile Mine, Lancaster County, S. C. (written in co-operation with Mr. A. Thies).

The Haile mine is situated 3 miles northeast of Kershaw in Lancaster county, S. C. It is the property of the Haile Gold Mining Company (New York office, 17 Maiden Lane), Capt A. Thies, superintendent and general manager.

This mine represents an example of gold-mining in its highest development in the South on large bodies of low-grade sulphuretted ore.

It is situated in the Carolina belt. The country is a siliceous hydro-muscovite and argillaceous schist striking N. 45° to 70° E., and dipping 55° to 85° N. W. The rock is impregnated with auriferous pyrite, free gold, and in places small quartz stringers. This is the mass that constitutes the ore-bodies, which are lenticular in shape. Their outline, however, does not necessarily conform with the strike and dip of the slates, but is determined rather by the degree of impregnation. The lenses are about 200 feet in length and 100 feet in maximum width. The pitch is 50° to 60° N. E., and the dip N. W. from 45° to nearly vertical. The country is intersected by a number of diabase dikes, from a few feet to 150 feet in width, striking across

FIG. 19.



Mines and Plant, Haile Gold Mining Co., Lancaster County, S. C. Scale, 1 inch = 400 feet.

A, Red Hill pits; B, Diabase dikes; C, clay dike; D, outlet of large reservoir; E, Bumalo pit; F, quartz vein; G, Beguelin mine; H, Haile pit; K, small reservoir; L, Chase Hill pits; 1, chlorination house; 2, roasting furnaces; 3, boiler house; 4, pump; 5, machine shop; 6, No. 2 shaft; 7, new shaft; 8, No. 3 shaft; 9, offices; 10, superintendent's residence; 11, mill; 12, concentration house; 13, boiler and engine; 14, crusher; 15, new Beguelin shaft; 16, Beguelin slope; 17, boiler house; 18, crusher; 19, flume; 20, mine railroad; 21, commissary; 22, church; 23, school; 24, village.

the slates at various angles, and in one instance (Beguelin mine) parallel with them. Where these dikes cross the ore-bodies they appear to have exerted, in some cases, an enriching influence on the ore. A short distance to the southeast of the main workings is the outcrop of a heavy quartz-vein (F., Fig. 19) from 10 to 12 feet thick, which strikes parallel to the slates; it is apparently barren. As explained above, the ore consists of pyritic slates, silicified in varying degrees, from soft sericitic slate to very hard hornstone. The slates of medium hardness are usually the richest; graphitic laminæ are also good indications. In the better grade of ore the pyrite exists in a finely-divided condition. Ore containing coarse sulphurets is generally of poor grade. The crucial test, however, of the value of the ore is the amount of free gold it contains, which is in direct proportion to that contained in the sulphurets, and is determined by daily panning. The ore at present delivered to the mill averages $4 per ton (assay-value), of which about one-third is free gold.* The percentage of sulphurets in the ore varies from 2 to 25 per cent.

The first work done at the Haile mine consisted of branch washing in 1829, which led afterwards to the discovery of gold on the hillsides. All work was open cutting until 1880, when underground mining was begun, and this is continued to the present time. Although visible coarse gold is now of rare occurrence, the mine has yielded some nuggets worth from $300 to $500 from the decomposed slates in the shallow open cuts.†

The first mill was a 5-stamp one, afterwards enlarged to 10, and in 1881 to 20. About 1884, a Blake dry crushing-mill was erected in connection with 20 Embrey tables.‡ This was soon abandoned, and the mine was worked in a dilatory way with the 20-stamp mill until 1888. During this time, and previously, many unsuccessful experiments for the treatment of sulphurets were made.§ In 1888, Mr. A. Thies took charge of

* Ores as low as $2.75 have been successfully milled.

† *First Annual Report on the Survey of South Carolina for* 1856, by O. M. Lieber, Columbia, S. C., 1858, p. 62.

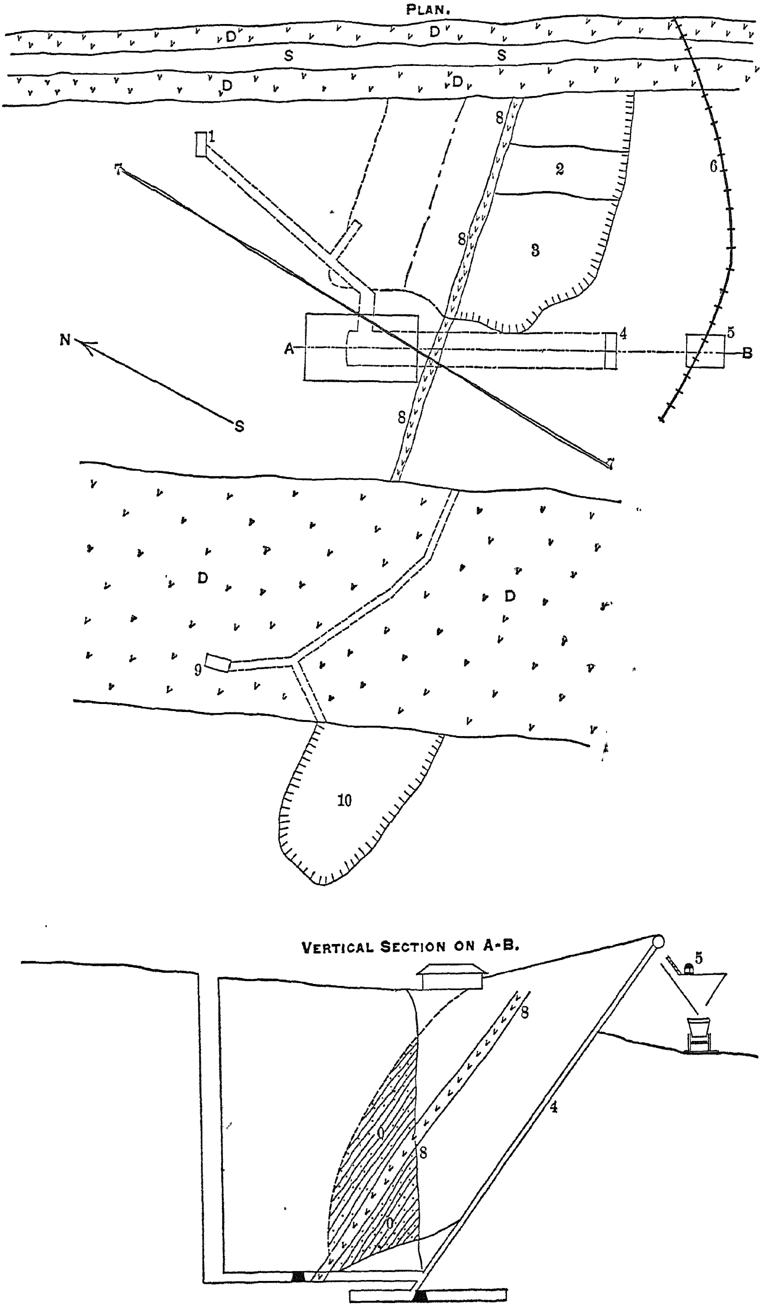
‡ "The Blake System of Fine Crushing and Its Economic Results," by T. A. Blake, *Trans.*, xvi., 753.

§ "Gold Mining in South Carolina," by E. G. Spilsbury, *Trans.*, xii., 99.

"Notes on the General Treatment of the Southern Gold Ores and Experiments in Matting Iron Sulphides," by E. G. Spilsbury, *Trans.*, xv., 767.

"Chlorination of Gold-Bearing Sulphides," by E. G. Spilsbury, *Trans.*, xvi., 359.

Fig. 20.



Beguelin Mine, Haile Gold Mining Company. Scale, 1 inch = 80 feet. D, diabase dikes; S, slate; O, ore-body; 1, new shaft; 2, pillar; 3, pit, 160 feet deep; 4, inclined shaft; 5, crusher and ore-bin; 6, mine railroad; 7, 22-inch diabase dike; 8, diabase dike, parallel to ore-body; 9, old shaft, 50 feet deep; 10, open-cut, 40 feet deep.

the Haile mine. He operated the 20-stamp mill until the mine was sufficiently developed to warrant a larger plant. At this time a 2-barrel chlorination-plant was added and increased later on to 3 barrels. In 1889, the Blake mill was changed to a 60-stamp, back-to-back mill, with 20 concentrators.

Description of Mine Workings.—The mines worked at present are the Cross (a continuation of the old Haile and Flint pits, H, Fig. 19, in depth) and the Beguelin (G, Fig. 19). The Bumalo, Red Hill and Chase Hill pits (E, A and L, Fig. 19) have not been worked for some time, although in the first there has been considerable underground work.

Work at the Cross mine was stopped in 1888, and all attention was concentrated on the Beguelin (formerly Blauvelt) mine. Fig. 20 gives a plan and vertical section of the open pits and some of the underground workings of this mine. The old workings consist of some shallow open pits and 3 perpendicular shafts, one 70 feet deep in ore, one 54 feet deep in the diabase dike (9, Fig. 20), and one 70 feet deep in the foot-wall slates on the southwest side of the dike (not shown). The first of these was transformed, from a depth of 60 feet downward, into an inclined shaft (4, Fig. 20), and sunk in the ore-body to a depth of 195 feet. This shaft was rigged with self-dumping skip, crusher and ore-bin situated over the railroad tracks which had been extended to the mine. At 60 feet a drift was run in a northeast direction until the diabase dike was reached. Meanwhile sinking was continued in the shaft to 120 feet. From this level drifts were run and connections were made with the 60-foot level, which prepared the ground between them for stoping. At 180 feet a similar drift was run to the dike and connections made with the upper levels in such a manner that the ore from the 60-foot would fall to the 180-foot level, and from there be hoisted to the surface. At 180 feet a drift was started in a southwesterly direction, encountering at 64 feet a dike 125 feet thick, through which the drift was continued to a distance of 600 feet from the shaft. At a depth of 70 feet a similar drift was run and the ore-body beyond the dike was prepared for stoping by connecting these two drifts by several raises. At the present day all ore on the west side of the dike has been stoped out to the 180-foot level. To the northeast of the shaft a considerable ore-body was still standing above the 60-

foot level. In order to extract this ore it became necessary to open the mine from the surface, and the open pit (3, Fig. 20) was started. The ground was stripped to a depth of 15 feet, and from that point on the ore, though lean, was used in the mill. At 60 feet a diabase dike (8, Fig. 20), lying parallel to the schistosity of the country was encountered in cross-cutting, and was at first believed to represent the hanging-wall. On cutting through it, however (a distance of 4 feet), it was found to merely divide the ore-body. Under the altered conditions it became necessary to sink a new shaft (1, Fig. 20) in the hanging-wall as an outlet for the ore and for pumping. This shaft was sunk to a depth of 165 feet; connections were made by cross-cuts with the present inclined shaft and everything prepared for taking out the shaft pillars, as well as the balance of the ore. This is the present condition of the mine. The maximum thickness of the ore-body at the Beguelin was 80 feet, and the best ore was found between the two large cross-dikes. A large amount of heavy sulphuretted ores is at present in sight.

Five hundred feet northeast of the Beguelin mine are several open pits known as Chase Hill (L, Fig. 19). The character of the ore at this point is somewhat different, being a banded, colored slate, barren of sulphurets, but carrying several gold-bearing quartz veinlets. Taken as a body it will not make ore.

To the northwest of the Beguelin are several ore-leads as yet unprospected.

The 60-stamp mill was run on Beguelin ores three years. The Cross mine was then reopened (1891). A plan of the Cross mine is given in Fig. 21, showing the open pits and present underground workings, as well as some of the abandoned ones. After the water had been pumped out, and the old Shaft No. 2 (*a*, Fig. 21), 200 feet deep, was fully secured, a cross-cut was driven in a northwesterly direction from the bottom, a distance of 25 feet. A drift (*f*, Fig. 21) was started from that point in a southwesterly direction, reaching ore at a distance of 75 feet from the cross-cut. This drift, on being continued 200 feet, encountered a dike 25 feet thick, which was cut through and the drift carried on for 100 feet more. The old workings (*d*, Fig. 21) were also continued through the dike, the drift (*e*, Fig. 21) on the 100-foot level being run 100

feet beyond it. Four upraises were driven between these two levels, two on each side of the dike, opening up 4 large stopes of ore. This ore ran low in sulphurets, but carried more free gold and furnished one-half of the quota to the mill. In order to work the ores below the 200-foot level a new shaft (*c*, Fig. 21) was sunk to a depth of 270 feet. A cross-cut was run from the bottom in a southwesterly direction for a distance of 75 feet; 15 feet from the shaft a drift (*h*, Fig. 21) parallel to the drift (*f*) on the 200-foot level was carried in a distance of 250 feet. The dike when encountered was 35 feet thick and no longer decomposed on the wall, as was the case in the upper level, but hard and solid. By upraises 4 more stopes were opened, and these furnish the present output of the mine. The ore was of a better grade in proximity to the dike on both sides.

The old workings (S, Fig. 21), which were continued from the Bumalo pit (B, Fig. 21) to a depth of 200 feet, but are now inaccessible, are at present being opened up by a diagonal drift from the 270-foot level (*h*, Fig. 21). Some time ago a northeast tunnel was driven from the Bumalo pit, at a depth of 50 feet and for a distance of 150 feet, to a diabase dike, 150 feet in thickness, and later continued through this. Drifts on the further side showed up only barren ground, but good ore was found from the mouth of the tunnel to the dike, being richest near the dike. It is expected to strike this ore-body at a depth of 270 feet with the drift above mentioned.*

So far as explorations have gone, 3 different lenses have been encountered: 1. The Bumalo, furthest northeast; 2. The Haile or middle lens; 3. A small lens 80 to 90 feet west of the Haile (outcrop under the new boiler-house, 17, Fig. 19).

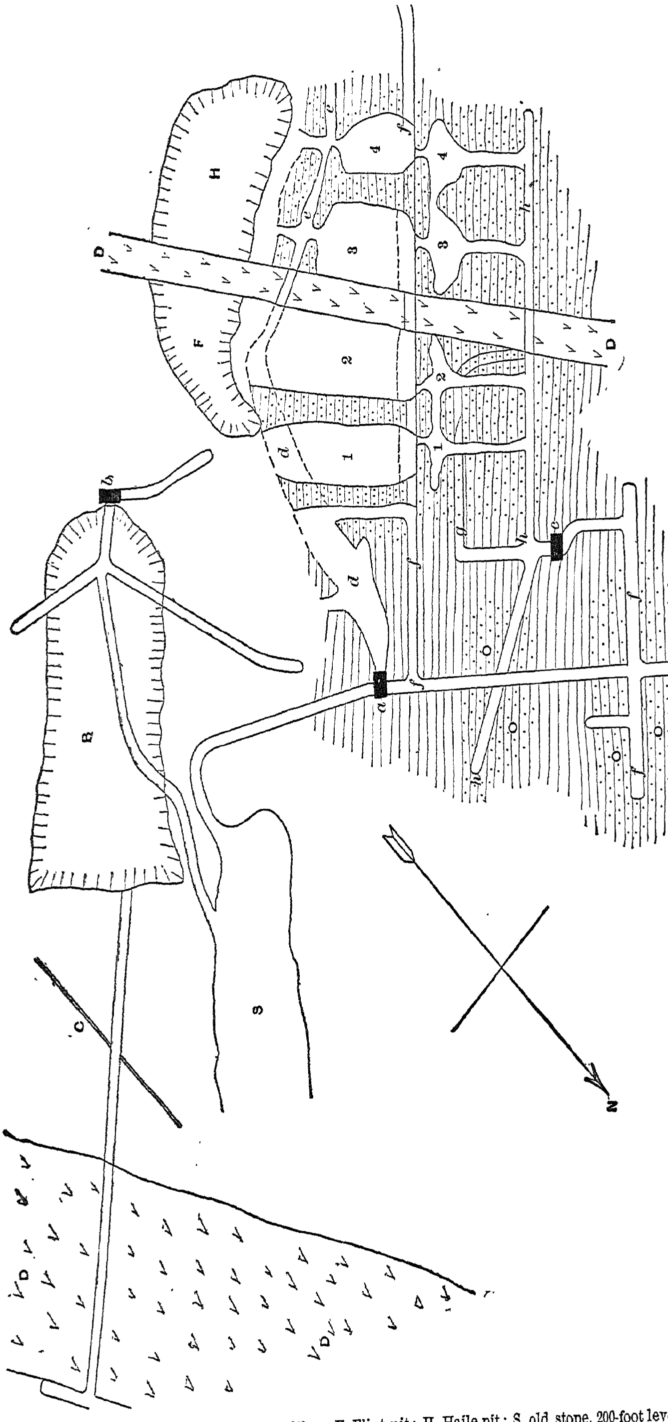
Red Hill (A, Fig. 19) consists of a number of open pits on the northwest side of the 150-foot dike, where ore was formerly mined to a depth of 60 feet. It is supposed to be in a line with the Haile lens.

The thickness of these lenses varies, reaching 100 feet in places, while at others, near the end of the lenses, it is only from 25 to 30 feet.

Method of Working.—The method of working these deposits is the pillar system (Pfeilerbau), illustrated in Fig. 22.

* Since the above was written the ore-body has been reached. It shows more heavily sulphuretted ore than any found in the Cross mine.

FIG. 21.

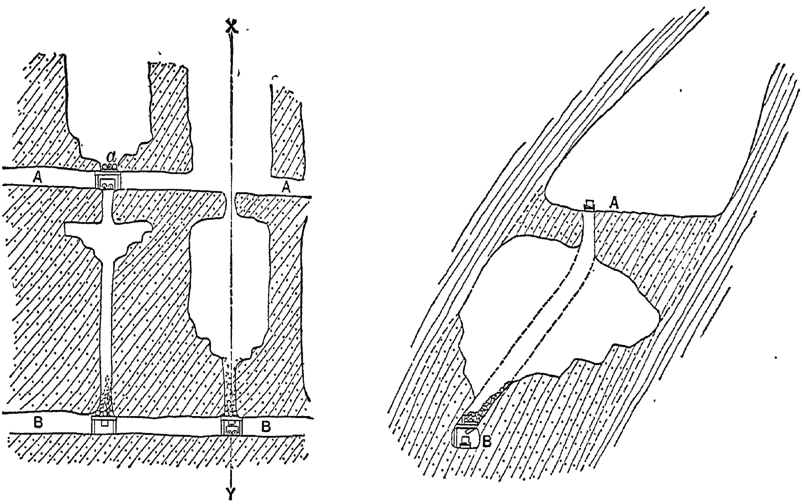


Plan of Cross Mine, Haile Gold Mining Company. Scale, 1 inch = 100 feet.

B, Bumalo pit; C, clay dike; D, diabase dikes; F, Flint pit; H, Haile pit; S, old stope, 200-foot level; *a*, No. 2 shaft; *b*, No. 3 shaft: *c*, new shaft; *d*, bottom of old stope, 160 feet rising to 100 feet; *e*, 100-foot level; *f*, 200-foot level; *g* and *h*, 270-foot level; O, ore-bodies; 1, 2, 3, 4, stopes.

The levels (8 by 7 feet) are run 70 to 100 feet apart, and nearer the hanging- than the foot-wall. At intervals of about 50 feet upraises are made, with a cross-section of 8 by 7 feet. These are carried forward at an inclination as near as possible to 45°. If necessary, the upper portion through the chain pillar left under each level is carried up vertically. This raise serves afterwards as a chute (mill-hole). Drifts are then run below this pillar until the limit of the stope in length (about

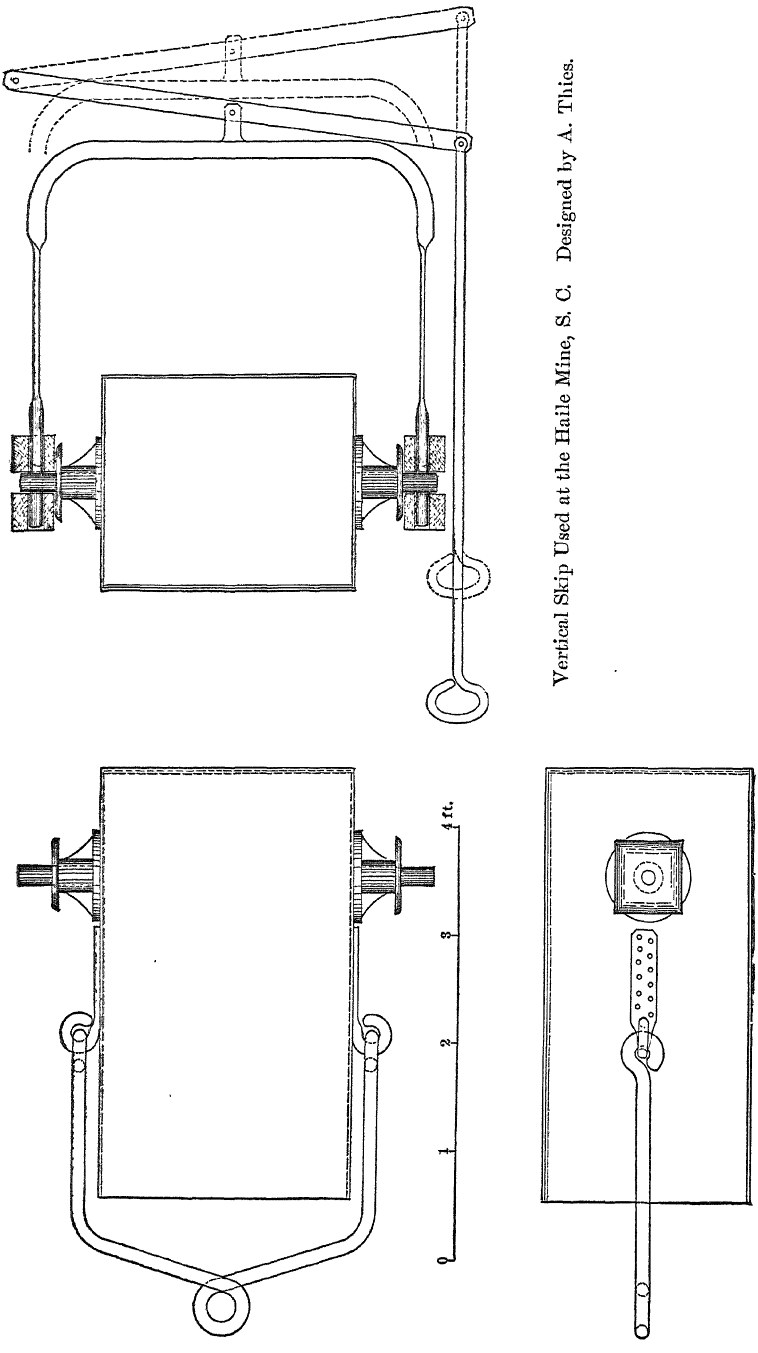
Fig. 22.



Vertical Section along Strike. Section on X-Y.
Method of Stoping at the Cross Mine. Scale, 1 inch = 60 feet.

30 to 40 feet in all) is reached, leaving a vertical pillar 15 to 20 feet in thickness between the stopes. The ground is then cut away between the foot- and hanging-wall, completely exposing as roof the bottom of the chain-pillar above, which is sprung in the shape of an arch, with its heavier toe in the foot-wall and a minimum thickness of 15 feet. This, as well as all other work in tight ground, is done by air-drills. Stoping is then carried downward by hand-drilling in circular steps, arranged in such a manner as to allow the ore to drop into the chute on blasting, without further handling. The angle of 45° given to the latter allows a steady flow of the material down the foot-wall without completely choking it. At the bottom of the chute is a rough grizzly (*a*) made of logs, which holds back the larger bowlders and prevents them from choking the

FIG. 23.



Vertical Skip Used at the Haile Mine, S. C. Designed by A. Thies.

smaller loading-pocket below. This grizzly is easily accessible from the drift, and the larger pieces of ore are here sledged. The loading-chute and grizzly are kept up as long as possible until the stope is finally broken through to the drift-level below, the ore being shoveled into cars. As far as possible, the pillars are left in poor ore, the diabase dike fulfilling this purpose admirably. No timber whatever is used, and although chambers 100 by 100 by 40 feet have been cut out, there seems to be no danger of a fall, the country-slate being very tough and self-supporting. The stopes from the 100- and 200-foot levels are connected with the surface by raises, so that at a future date the worked-out stopes can be filled from the surface and the ore in the pillars, *i.e.*, what is left toward the hanging-wall, can be taken out.

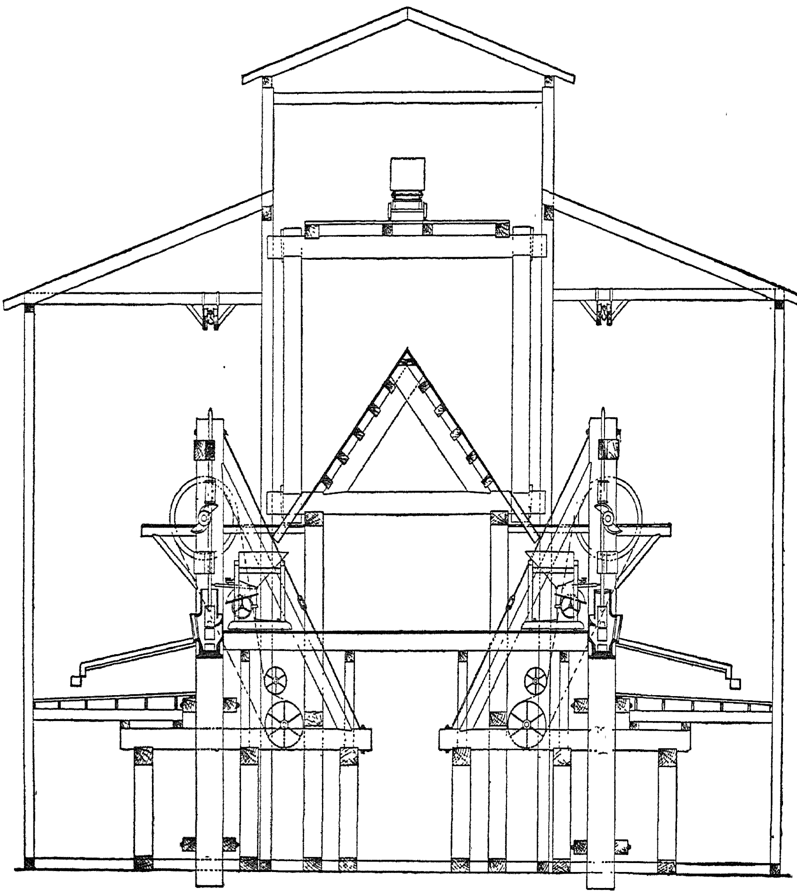
Blasting is done with 40-per-cent. Hercules powder. One-inch steel is used for both hand- and machine-work. The number of air-drills is limited by the size of the compressor—an Ingersoll machine, with 3-drill capacity. The ore is carried from the loading-chutes to the shafts in sheet-iron cars of $\frac{3}{4}$-ton capacity, running on 18-inch gauge track. At No. 2 Shaft (7 $\times$ 12 feet, single compartment) they are hoisted by cage, with automatic safety-catch. The new shaft is 6 $\times$ 14 feet, double compartment, and the ore is raised by a novel skip designed by Mr. Thies (Fig. 23). The body of the skip, made of sheet-iron, has two projecting lugs riveted to it below the centre of gravity and the bail is lugged one inch from the vertical centre-line.

Each lug runs between a pair of yellow-pine guides set 2 inches apart. When the skip is raised above the landing-chute two iron pins are thrown across the openings between each set of guides. The skip is dropped down on these and the ore is dumped into a loading-chute placed on the heavier side of the skip. The skip is raised and righted by the bail, the iron pins are withdrawn by the lander, and the skip descends. The operation is rapid and simple and the cost of the device is light. The mine is not wet, a No. 9 Cameron pump easily handling the water.

Milling.—The ore is crushed to $1\frac{1}{2}$-inch size in a 10 $\times$ 20-inch Blake crusher at the Beguelin, and a 7 $\times$ 10-inch crusher at the Cross mine, and is stored at both places in bins of 30

tons capacity. The broken ore is hauled to the mill in narrow-gauge, bottom-dumping cars, holding 3 tons; 8 cars are run to the trip. The mill bin has a capacity of 300 tons, and is so arranged that every stamp can be supplied separately with ore, as, owing to the different character of the ores at the Beguelin

FIG. 24.



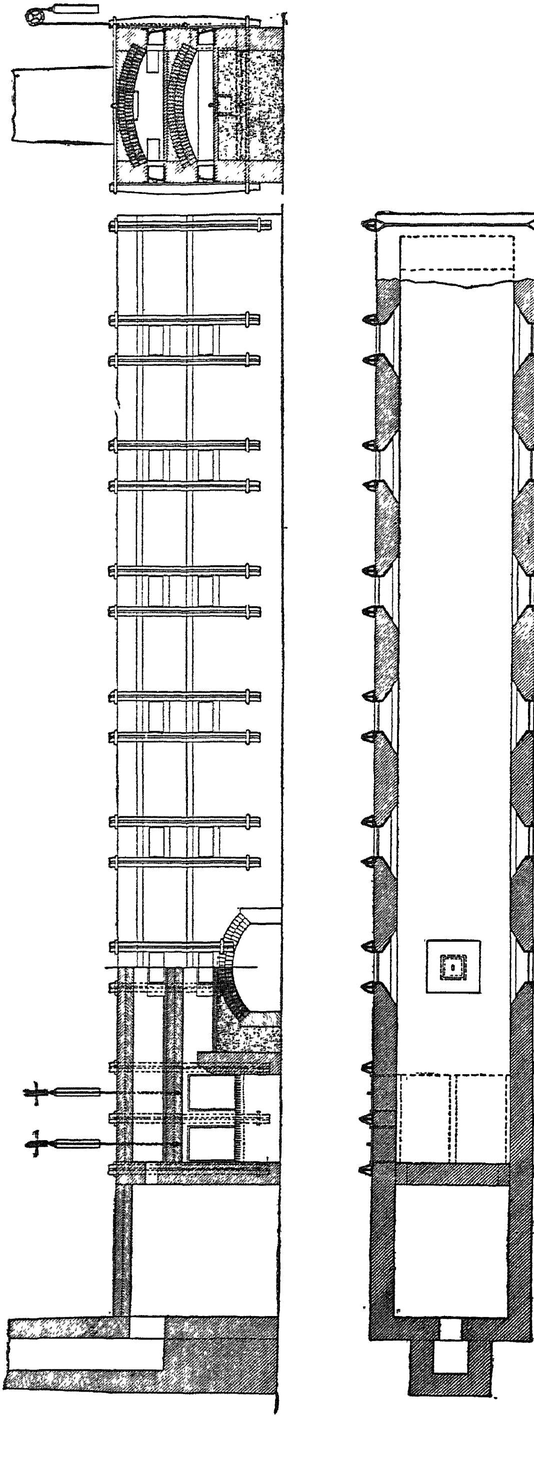
Vertical Cross-section of 60-stamp Mill at the Haile Gold Mine.

and the Haile, it is treated in separate batteries. A hinged plate, not shown in drawing (Fig. 24), is for this purpose hung at the apex of the bin floor. A vertical cross-section of the mill is shown in Fig. 24. Two vertical sections of the battery are shown in Fig. 16, under the Reimer mine (see p. 755).

The mill is a 60-stamp back-to-back one, 30 on each side, built by the Mecklenburg Iron Works, of Charlotte, N. C.

The ore is fed by Hendy self-feeders. The weight of the stamps is 750 pounds; chilled iron shoes and dies are used; the stamps drop 6 inches, 86 times per minute, in the order 1, 3, 2, 5, 4. The crushing capacity is 2 tons to the stamp in 24 hours. The screens are 30-mesh brass, made of No. 20 wire; these work well if no cyanide is used in the battery. The average height of discharge is 6 inches. Amalgamation is accomplished: 1. In the mortar by a curved front plate attached by means of a wooden chuck block to the lip of the mortar, immediately below the discharge; it is held in position by bolts and can be rapidly and easily removed. It presents an amalgamation-surface of 1.75 square feet and is made of No. 7 silver-plated sheet-copper. The gold being very fine, its accumulation in the mortar between the dies is insignificant, and the mortar is seldom cleaned out. 2. On the outside plate, made of No. 12 silvered copper-sheet, and presenting an amalgamation-surface of 32 square feet; they are the full width of the mortar and are arranged in four steps, each 2 feet in length, and overlapping the next by 1 inch, the inclination being 2 inches in 1 foot. They are fastened directly to the battery, the tremor caused hereby being considered beneficial to amalgamation. These plates are interchangeable; whenever the upper plate becomes hard and unfit for amalgamation, it is interchanged with one of the lower plates, thus giving in rotation to each plate a position at the head of the table. Each battery is provided at the screen-discharge with an impact-plate, not only for amalgamation, but to retard the velocity of the pulp. They are cleaned from verdigris with a weak solution of cyanide, and a little potash is sometimes fed into the battery. Phosphate of sodium is used in the mill to keep the quicksilver bright and lively. It has been found expedient to remove the inside plates every 24 hours; as duplicate plates are kept on hand, no delay occurs while they are being cleaned. The amalgam from these, which is collected and weighed daily, forms an excellent indication of the value of the ore milled. The amalgam is removed from the outside plates whenever it is necessary. A regular clean-up is made only once a month. About one-third of the gold is saved on the inside plates. The fineness of the mill gold is 880. The average amount of water used per stamp is 3½ gallons a minute; and the average con-

Fig. 25.



Double-Hearth Roasting Furnace; Chlorination Plant; Haile Gold Mine. Dimensions of Hearth, 5 feet wide by 40 feet long.

sumption of quicksilver is 0.35 ounce per ton of ore. The wear of shoes and dies is 1.3 pounds per ton of ore stamped. As a lubricant for the cams, molasses thickened with flour is used and gives excellent results. It is contemplated to add 12 additional square feet to the outside amalgamation surface of each battery; this will be arranged by a drop-system of three plates, the pulp discharging from one to the other before it enters the main launder.*

The pulp is carried to the concentrators in launders lined with riffles. No attempt at sizing the pulp is made, but the ores from the Beguelin and Cross mines, owing to the difference in contents of sulphurets, are concentrated separately. The Cross ore averages about 2 per cent., the Beguelin running from about 7 to 25 per cent. sulphurets. They are milled separately in the proportion of $\frac{4}{10}$ Beguelin and $\frac{6}{10}$ Cross, so as to obtain an average of 7 to 8 per cent. sulphurets from the total ore milled. The concentration is done on 20 Embrey tables (4 by 12 feet), with smooth rubber belts which are set at an inclination of $2\frac{5}{8}$ inches and travel 5 feet per minute, receiving at the same time 192 percussions. The concentrates contain 90 per cent. pyrite, which is pure sulphide of iron with occasional small traces of arsenic. The average value of these concentrates is $25 to $35 per ton.

Chlorination.—The concentrates are hauled on the mine-railway to the chlorination-plant. They are roasted in two double-hearth reverberatory (see Fig. 25) and one revolving pan furnace,† the sulphur being reduced from about 43 to as low as $\frac{1}{4}$ per cent., and the value of the material being increased by $\frac{1}{3}$.‡ Each double hearth furnace is worked by two men to a shift of 12 hours, the output being 2 tons of roasted concentrates per 24 hours for each furnace. The revolving pan furnace is worked by three men per 24 hours, with the same output as the double hearth. The fumes from these furnaces carry off into the air the equivalent of 13 tons of 50-per-cent. sulphuric acid. The management have investigated the erection of lead cham-

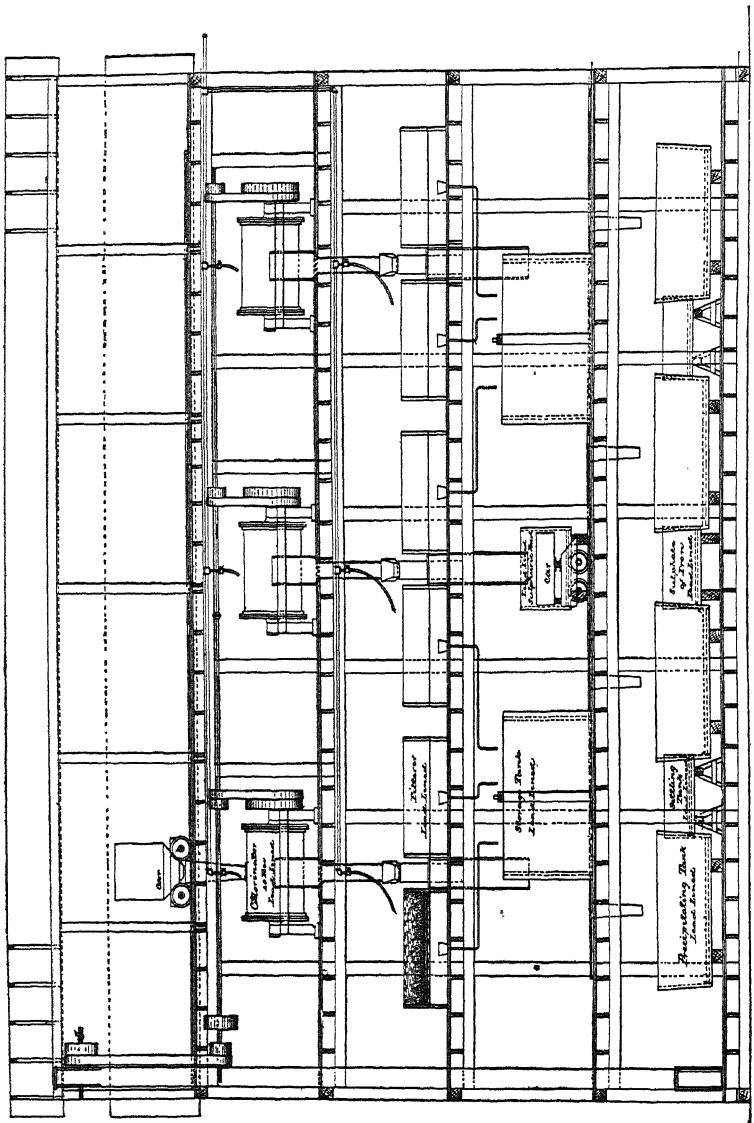
* This has been done.

† For full description of this see paper by W. B. Phillips, "Chlorination of Low-Grade Auriferous Sulphides," *Trans.*, xvii., 313.

‡ A practical test for dead roasting is made by the furnace-man by dipping a bright iron rod into a boiled aqueous solution of the roasted ore.

bers, but so far have not considered such an installation to their advantage. The Spence furnace has been tried at the Haile, without success.* The roasted ore after cooling is elevated to

FIG. 26.



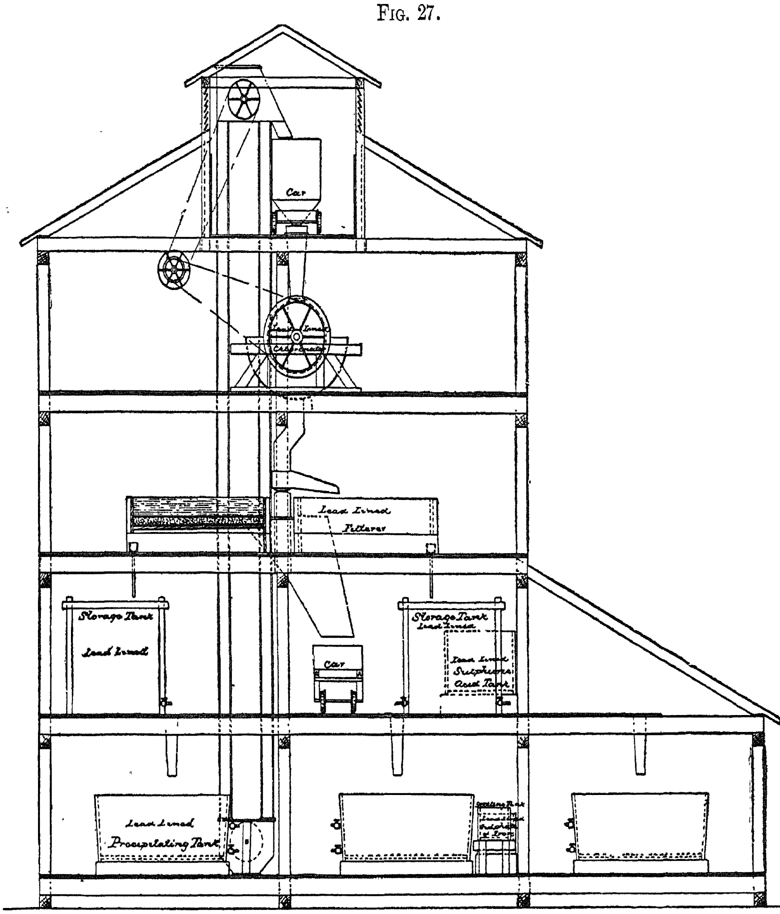
Chlorination Plant at the Haile Gold Mine. Vertical Longitudinal Section.

the top floor of the chlorination-house, 32 feet high. This consists of a four-story frame building containing 3 chlorination-

* See paper by W. B. Phillips, "The Thies Process of Treating Low-Grade Auriferous Sulphides at the Haile Gold Mine, Lancaster Co., S. C.," *Trans.*, xix., 601.

barrels, 11 filtering-tanks, 2 storage-tanks, and 13 precipitating-vats (see Figs. 26, 27). The ore is charged by cars holding 1 ton, through a hopper into the chlorination-barrels (see Fig. 28). The barrel is 60 inches long by 42 inches in diameter, made of cast-iron and lead lined (12 pounds of lead to the square foot).

FIG. 27.

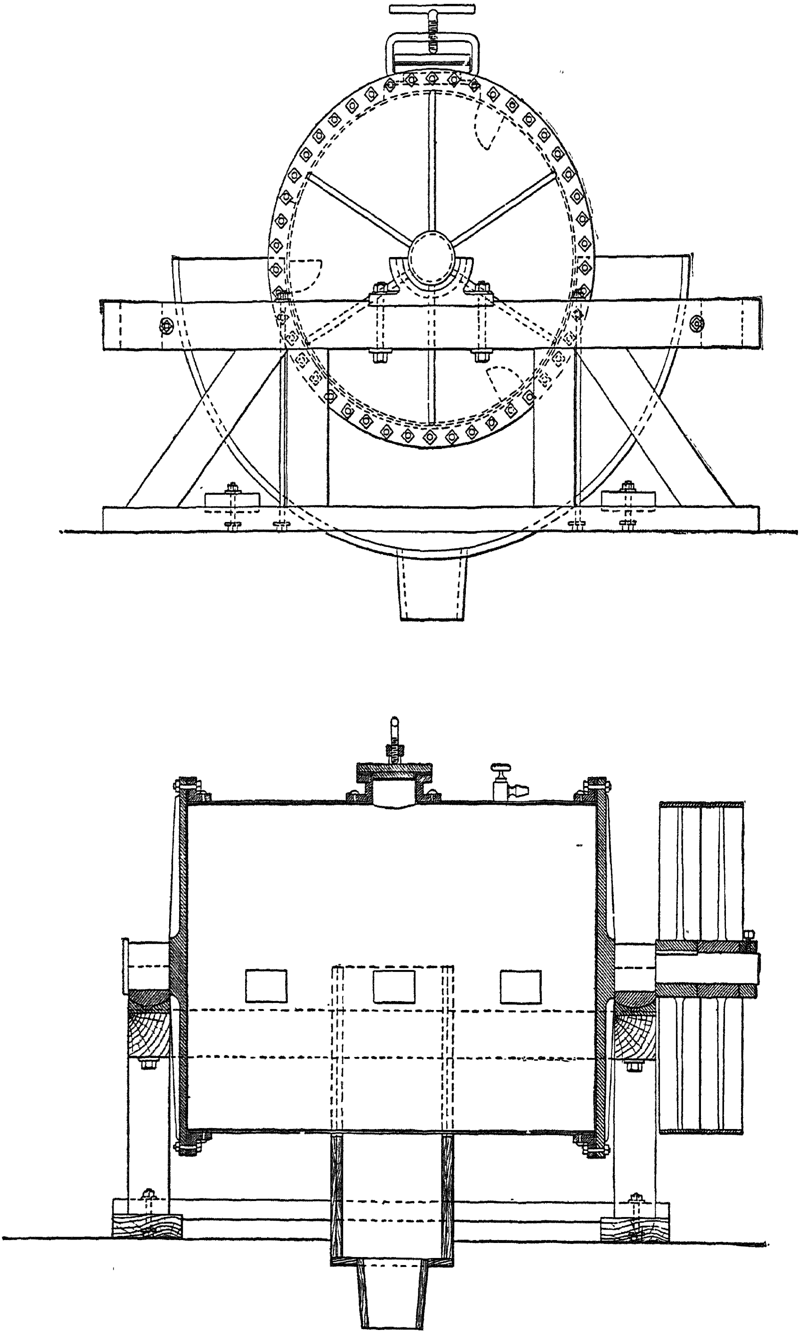


Chlorination Plant at the Haile Gold Mine. Vertical Cross-section.

It also contains a lead-valve in order to ascertain whether the necessary amount of free chlorine is present. (The use of this valve is unnecessary after the character of the ore becomes known.)

The full charge consists of 120 gallons of water (to make an easily-flowing pulp), from 8 to 11 pounds of bleaching-powder, then the ore, and finally 12 to 15 pounds of sulphuric acid.

FIG. 28.



Chlorination Barrel, Haile Gold Mine. Inside dimensions: Diameter, 42 inches; length, 60 inches.

The barrel is hermetically closed and revolves for about 3 hours, at the rate of 15 to 18 revolutions per minute. (A 5 horse-power engine performs this work and also the elevating of the ore.) The barrel is then inverted, opened and discharged through a lead-lined semi-circle in the floor to a filter on the floor below. There are 4 lead-lined filters to each barrel, their dimensions being 6 by 8 feet by 18 inches deep in front and 17 inches in back. The bottom is covered with mineraline* tiles; 12 by 12 inches by 1 inch thick, perforated and having $\frac{1}{2}$-inch gutters underneath; on top of these is placed a rack of $1\frac{1}{4}$-inch wooden slats, 4 inches high and 8 inches apart; the first layer above the tiles consists of 4 inches of coarse quartz pebbles ($\frac{1}{8}$ to $\frac{1}{2}$ inch in size), and this is covered by from 1 to 2 inches of ordinary clean sand. Before emptying the contents of the barrel, the filter is flooded with water to the level of the top of the filter-bed, to act as a cushion. Then the original solution is passed through, striking on a float to prevent breaking the filter-bed. The ore-pulp is washed twice with clean water; the first time enough is added to stand 4 inches above the surface of the pulp, and the second time the tank is entirely filled. This amount has been found sufficient to thoroughly remove all traces of chloride of gold from the pulp (tests are made with $FeSO_4$). The filtered solutions are stored in two stock tanks on the second floor, and are drawn off from these into the precipitating tanks as required. The latter are 8 feet in diameter and 3 feet high, made of wood, the interior coated with asphalt. They are provided with three outlets, the upper one 18 inches from the top, the middle one 1 inch above the bottom and the lowest one in the jamb. The gold is precipitated in the metallic state with an excess of fresh ferrous sulphate, made in a small lead-lined tank. In warmer weather 48 hours suffices for settling, and in colder weather from 3 to 4 days. The supernatant liquor is drawn off through the two upper outlets, opened one after the other (in order to prevent any stirring of the precipitates), and passed through a box filled with sawdust to catch any precipitate. The gold precipitate is drawn from the tanks through the jamb opening into a small lead-lined settling tank 2 by 2 by 4 feet. After standing 24 hours the supernatant liquor

* A melted mixture of sulphur and quartz.

is siphoned off, and the precipitate filtered on paper. This is dried and mixed with about half its weight of borax and soda in almost equal proportions. Should iron salts be present, a little quartz-sand is added. It is melted in graphite crucibles and cast into ingots of about 990 fineness. The whole operation is so simple that the most ordinary laborer can acquire the mechanical knowledge in a day. The repairs are practically *nil.**

Labor, Costs, etc.—Some of the figures of costs of labor and working at the Haile mine are given below. For private business reasons it is impossible to give these as fully as we should like to.

Mines.—Cost of Labor:

	Per day.
Holders,	$ .90
Strikers,	1.10
Machine-runners,	1.25

255 cords of wood, at $1.50, burned per month.

Mill (2 shifts of 12 hours each).—Distribution and Cost of Labor:

One superintendent,	$
" laborer (amalgamation) per shift,	2.50
" " " "	1.00
" " (concentration) "	1.25
" " " "	.80
" fireman "	1.00
" engineer per day shift,	1.75
" " per night shift, at,	1.50

150 cords of wood at $1.50, used per month.

Repairs (wear of shoes and dies, etc.), 4 cents per ton of ore.

Roasting and Chlorination.—Distribution and Cost of Labor:

Roasting furnaces, producing 6 tons of roasted concentrates per 24 hours: Six men, day shift, at,	$1.00
Five men, night shift, at,	1.00

3 cords of wood, at $1.50, used per 24 hours.

Chlorination, 1 shift of 12 hours working 6 tons of roasted concentrates:

Two men at,	$1.00
One man at,	1.25

Cost of roasting per ton of roasted concentrates:

Labor,	$1.83
Fuel,	.75
	$2.58

* The Thies chlorination-process has lately been described in detail by T. K. Rose, in his *Metallurgy of Gold*, C. Griffin & Co., London, 1894.

Cost of chlorinating 1 ton of roasted concentrates:

Labor,	$.50
Foreman,	.20
Power,	.12
Sulphuric acid for $FeSO_4$,	.06
11 pounds of bleaching powder, at 2½ cents,	.27½
15 pounds of sulphuric acid, at 1 cent,	.15
Wear and tear,	.10
Superintendence,	.05
	$1.45½

Cost of roasting and chlorination per ton of raw concentrates,	$3.02
Cost of roasting and chlorination per ton of ore mined,	.19

Percentages of Extraction:

Mill:	Tailings from concentrators,	85 to 90 cents.
	Showing a yield of,	75 to 80 per cent.
Chlorination:	Tailings as high as,	$1.50
	Average yield,	94 per cent.

VII. Conclusions.

The figures taken from the Census Report of 1890 (see p. 688), which show the sum of about $535,000 as the cost of producing $318,000 of Southern bullion during the year 1889, are certainly discouraging, especially when considered in combination with the capital stated as therein invested ($5,900,000). If this poor showing were not due to a large proportion of misdirected efforts, the case would seem hopeless. Bonanzas have not been found in the South, and probably never will be, unless we except rich pockets of limited extent, which for a time might prove to be such to the individual operator or tributer. The Western saying that "A good gold-mine is one which will pay dividends under poor management," would exclude all Southern gold-mines from even this distinction. There are, however, a few mines in the South, notably the Haile and the Franklin, which, under able management, fully conversant with all the requirements and exigencies of the case, have been developed into remunerative business enterprises. The wide distribution and variety of the auriferous deposits throughout the South do not preclude the possibility of these mines serving as examples for a larger number of operations, instead of being isolated cases as at present. We have attempted to give a specific description of the ore-deposits and the methods of mining and milling. It may be in place here to give a brief general discussion of these and other points relating to gold-mining in the South.

By far the greater portion of the gold that has been produced in the South was derived from placer-mining, including bottom- and side-hill gravels, as well as auriferous saprolites and decomposed vein-outcrops. From such deposits the cream has been worked off, and what remains are the old gravel-heaps, and such virgin ground as in the earlier days proved inaccessible to water, and unprofitable for primitive methods, or was overlooked by the prospectors. Of the latter class, the Crawford mine, above described, is an example. Although the earlier prospecting for gravel was carried on in a thorough manner, there were no doubt large plantations on which such work, especially in the fertile bottoms, was not countenanced. It is also probable that deeper-lying gravel-channels, of which there are no indications on the surface, remain to be exploited, as, for instance, in the South Mountain and Dahlonega districts. The installation of pumps (or where these have been unsuccessfully used, the erection of improved or more economic plants), as well as more thorough and extensive surveys for ditch-lines, may open up much ground which was formerly inaccessible to water. Hydraulicking under direct pressure from a pump may in many cases be feasible, and may prove more economical as far as plant is concerned.

Bottom-gravel mines were operated in the earlier days almost entirely by pitting, draining the excavations with water-wheels, and raising the gravel by hand to rockers and sluice-boxes, the tailings being left in large heaps. This work was often done in an unsystematic manner; portions of the ground could not be worked at all; and, in general, only the richest gravel received attention, the over-lay and the bed-rock being neglected entirely. Some of these gravel-heaps have frequently been reworked, in one case (on the Mills property, N. C.) as often as seven times. The additional gold obtained in these operations was partially due to the incompleteness of the preceding washings, as well as to the subsequent further disintegration of vein-quartz, carrying free gold and sulphurets. A number of these old bottom-placers may warrant a remunerative reworking on a large scale, either by the use of giants and bed-rock sluices, when sufficient fall is available, or where the latter is not the case (a common feature in the South), by the application of the hydraulic gravel-elevator.

Virgin placer-deposits also exist, which, on account of the low grade of gravel, or the great depth of the over-lay, could not be profitably worked by the more primitive methods. For such, the above appliances may also give a solution. The Southern gravel-deposits are far less extensive than those of California and New Zealand, and therefore as low a grade of gravel cannot be worked, although the South has cheaper labor in its favor. Systematic work has rarely been pursued, and records of such work have not been kept. For this reason, as well as on account of the unequal concentration of the gold in the bottom and the varying working conditions met with, it is impossible to give limiting values per cubic yard to guide operations in the future. For the same reasons, preliminary testing will be difficult, especially in worked ground.

In general, it may be said that the great extent of the rock-decomposition in the South (often from 25 to 100 feet in depth), and the easy disintegration of the same, has resulted in a greater concentration of gold in the gravel, considering the richness of the ore-bodies in place, than in many other gold-fields.

The auriferous saprolites and decomposed vein-matter have been most extensively worked in the Dahlonega district. Here the decomposed material, in which gold from the eroded vein-matter is more or less concentrated, has to a great extent been worked down to the harder rock. In the Dahlonega method of working, everything seems to have tended to the simplification of the process and plant, with the object of milling as large an amount of low-grade material as is possible with economy in labor and plant, irrespective of close extraction. Both on account of the greatly impoverished material and its increasing unfitness for disintegration with the giant, a limit to this method of mining must ultimately be reached here. The ore-bodies continue in depth, and should open up a probably more productive field in deep mining, with less loss of gold and more economical output.

Although the Southern gold-field has been known and worked since the beginning of the century, it has not had the benefit of such thorough and systematic vein-prospecting as most of the later-discovered fields. It was already a well settled farming country, generally owned in large plantation-tracts, when gold was first sought after; and such lands as were unoccupied

were the property of the State governments, which did not offer special privileges and inducements to the development of the mining industry. Hence, the Western system of mineral lands and mining claims did not exist, and the field was not opened to the individual professional prospector. The same condition practically exists to-day. It is difficult to make satisfactory arrangements with the property-holders for prospecting; and propositions for such work from outsiders are, as a rule, regarded with suspicion. Even the larger tracts owned at present by mining companies have not been prospected to any extent. A notable exception to this is the development-work carried on by the Yonah Land and Mining Company in Georgia. If this example were followed by other mining corporations whose acreage runs into the thousands, while their operations are limited to a few square rods, it would greatly help to develop the possible gold resources of the South, in the direction of new discoveries. We do not, however, wish to give the impression that larger and more valuable ore-deposits than those already exploited are still to be found; the more easily recognizable and richer outcrops have been worked over, and in any case such finds as may be made will probably present no new features.

In general, the abandoned mines present the same features as those that are working. Judging from some of the older reports (Silliman, Rogers, Emmons, etc.), the surface ores of these mines were very rich, due partially to local concentration near the surface from the eroded portions of the vein, and in other cases perhaps to pockets and shoots of limited extent and depth. In the earlier days, few of the veins were worked below the water level; the abandonment of these older mines cannot, however, always be laid to the appearance of unworkable sulphurets. In the sulphuretted ores worked to-day from 20 to 60 per cent. of the gold is free, and in many of the earlier mines, where rich ores occurred in continuous shoots, these were followed down far below the water level and the free gold which they contained was obtained by simple amalgamation, as for instance at the Gold Hill mine in North Carolina; where the workings extended to 740 feet in depth. The more plausible reasons for the abandonment of the so-called *rich* Southern gold-mines may be attributed to the pinching out of

the ore-shoots outcropping at the surface or a diminution in the assay value of the ore. It is probable that the more expensive and difficult operations at such depths precluded the further search for other ore-bodies below the water-level. It must also be remembered that as early as 1840 at least partially successful attempts were made to work sulphurets.

In many of the mines, however, the ore-bodies were of low grade, though sometimes of large extent, and the small extraction of the free gold in the sulphuretted ores did not permit of a profitable continuation of the work. As in all mining regions, many other so-called plausible reasons are given for the abandonment of the mines, as, for instance, mismanagement, disputes among the owners, etc. To test their value an examination of the mines is absolutely necessary. A conclusive opinion is, however, in most cases impossible, even after the mine has been pumped out and examined, on account of the poor condition of the workings and the at best limited development of ore-bodies. In order to determine the probable value of the mines the prospective buyer must, with few exceptions, bear the cost of the necessary exploratory development, which expenditure is in a way speculative. A great number of the properties are held at prohibitory figures, and arrangements for satisfactory examination under option or otherwise cannot be made, traditional merit and output being considered a sufficient proof of value by the owners.

In low-grade, highly-sulphuretted ore-bodies, assays may give a fair indication of the value of the ore, if the samples be fairly taken; but a test on a larger scale at one of the experimental chlorination-plants,* in cases where it is intended to subsequently adopt the chlorination process, would be much more conclusive.

On higher-grade, free-milling ores, however, assays, even if taken with care, will be of little value; the results will, in fact, often be misleading. In such cases a mill-test is imperative, which can generally be made either in the mill at the mine itself, at some neighboring mill or at test-mills especially operated for this purpose.†

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* Captain A. Thies, Haile mine, S. C., and Mecklenburg Iron Works, Charlotte, N. C.

† R. Eames, Jr., Salisbury, N. C.; Mecklenburg Iron Works, Charlotte, N. C.;

The most feasible propositions in the South appear to be the working of larger low-grade ore-bodies. Rich veins have been, as a rule, pocketed and of small extent, more suited to the operations of tributers or small land-owners, with the help, perhaps, of the wooden stamp-mill. It is a well-known fact that pocketed or uncertain deposits of gold, as well as of all other ores, cannot be worked systematically by larger companies, with an extensive plant, and must be left to the individual miner, whose personal success pays his daily wages, and to whom an occasional strike is an inducement for continuous work.

Systematic work can only be pursued where the ore-bodies are large and continuous enough to warrant the establishment of a regular plant for mining, milling and reduction of the ores. The question of quantity means more than that of quality, so long as the latter does not fall below a certain limit.

Among such may be classed the wide lenticular bodies of auriferous and pyritic slates, as at the Haile and Russell mines, and the persistent and continuous quartz-veins of sufficient width, such as at the Reimer and Capps mines. The more continuous and stronger ore-leads of the Dahlonega type may also be included here, such as the Lockhart and Franklin mines, which are at present being worked as deep mines, as well as those which have so far been worked by hydraulicking, like the Hand, Singleton, Findley, etc., mines.

In some localities smaller or irregular quartz-veins lying close together have been worked separately; it may prove feasible to mine these together as a body of low-grade ore, especially where the intervening and adjoining country-rock is to some degree auriferous, as at the Rocky River mine.

Such ores as are alluded to may be said to average between $3 and $7 per ton. There are exceptional cases of richer ore-bodies which have shown considerable continuity, as, for instance, at the Phœnix mine; but here, as is usual, the quantity of the ore decreases proportionately with the quality.

Almost without exception, a profitable extraction from Southern gold-ores can only be attained by supplementing amalgamation with concentration of the sulphurets and by subse-

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The Hand & Barlow United Gold Mines and Hydraulic Works of Georgia, Dahlonega, Ga.

quent treatment of the latter. The practically universal adoption of the stamp-mill in the South verifies, as in other gold-mining regions, its more general applicability for crushing compared with other machinery. The two types of stamp-mills more especially characteristic of the South, each having its own field of action, have been described on pages 745 and 756. The milling practice varies greatly, as might be expected from the extremely variable character of the ores.

All of the Southern ores contain at least a portion of their gold in the free state, and excepting where other ingredients offer serious obstacles, or where a smelting process is intended, concentration is best preceded by amalgamation, so as to obtain the free gold as soon as possible and not endanger it to loss in subsequent treatment. Especially where the sulphurets are coarse and the crushing is not fine, a preliminary sizing in hydraulic classifiers and spitz-kasten, and the treatment of each size on a separate vanning-machine, is advisable. There has been a tendency to overcrowd these machines in the South; a saving of original cost here is but poor economy. It would seldom be advisable to use less than two 4-foot belts to every five stamps. The degree of concentration (cleanness of the concentrates) must depend upon the ratio between the cost of subsequent treatment per ton, on the one hand, and a greater loss in tailings occasioned by close concentration on the other; the cost of concentration itself being practically the same in either extreme. For the economical treatment of the concentrates, chlorination by the Thies process furnishes, in almost all cases, a ready solution. The process is a simple one, and is not patented; the cost of plant is comparatively small; and the percentage of extraction is high (94 to 97 per cent.); it has been in active operation on a continuous working-scale at the two most successful mines in the South (Haile and Franklin mines). The presence of copper is objectionable in this process, as it increases the consumption of chemicals, and if in too large a quantity it may preclude the adoption of the process. At the Phœnix mine, N. C., ores running as high as 3 per cent. copper were, however, successfully treated. Ingredients which make dead-roasting difficult may also add to the cost of the process.

Sulphuretted ores assaying only $3 per ton, when existing in extensive bodies, so as to permit operations on a large scale,

other conditions being favorable, can be worked at a profit by the application of this process.

Should concentration, on account of the too finely divided condition of the gold and sulphurets, prove impossible without a heavy loss in tailings, the cyanide, bromination or Swedish chlorination process might prove of value for a direct treatment of the ore; or the ore might be treated in bulk by the modification of the Thies process in use at Deadwood, Dakota; or by the Thies process proper on an enlarged scale, using, if necessary, closed filtering-tanks under pressure. In all of the above previous roasting is necessary, excepting perhaps in the cyanide. Attempts with the latter have so far been unsuccessful. It will be of interest to watch the outcome of the plant proposed at the Russell mine, N. C. Lack of success in the use of cyanide cannot always be laid to its lack of applicability; it certainly has, however, this disadvantage, that it requires a careful experimental trial, best made on a large scale and therefore expensive, as well as a continuous supervision afterwards by an experienced chemist, together with more or less skilled assistance, which is a requirement not always conformable with Southern conditions.

A small class of the Southern ores, referring particularly to those containing lead, copper and zinc, will have to be treated by smelting. A smelting-plant would, however, only prove a financial success under the concerted action of all—or, at least, most—of the mines producing such ores, a state of affairs which, under the present condition of gold-mining in the South, seems difficult to imagine. Several attempts have been made to gain this end, but have not been successful.

Taken as a whole, the gold-ores of the Southern Appalachians present no greater difficulties of treatment than those of other fields, the distinguishing feature being, perhaps, their large variety, which makes a close study of each separate ore-body necessary.

As to the cost of labor in the South, it may be said that while it is low compared to that of the Western mining countries, and unskilled labor can be obtained at especially slight cost, skilled labor commands about the same wages as throughout the East. The so-called cheap skilled labor of the South is a misnomer. It will be found here, as in other places, that the

laborer is worthy of his hire. Some difficulty may be experienced in obtaining suitable labor, especially in those districts where no active mining work has been going on. In general, there are no mining camps, in the Western sense of the word, and hence no regular mining population that might otherwise engender a more active mining spirit.

Among the facilities for operations in the South are the climate, which permits continuous working throughout the year; the accessibility of the mines to railroad lines, and their comparative proximity to investing Eastern capital. Lumber, timber and cord-wood can be obtained at very low cost. Mining supplies and machinery are furnished from several central points in the field (Salisbury, Charlotte, Dahlonega, Atlanta). Water-power—in most cases, however, undeveloped—is abundant throughout a great portion of the mining-belt. Should a revival in mining favor a development of properties in groups, central electric-power distribution-plants would in most districts be practicable.

Gold-mining in the South has its favorable features, which should facilitate the economic working of the ore-deposits as legitimate business undertakings, with close and intelligent management. A considerable number of properties are at least worthy of investigation, and to the best of our belief, such investigations will disclose remunerative working-opportunities, and will ultimately lead to a reasonable revival of gold-mining in the South. Examinations would be greatly stimulated by more disinterested co-operation and reasonable demands of the mine-owners, ultimately to their benefit. Speculative investments in the Southern gold-mines have had their day, and unsophisticated capital is becoming rare.

Postscript.

The following later information is added to complete the foregoing paper:

Sam Christian Mine (p. 700).—In pumping water from the Yadkin river for this mine, the late Sam Christian Co., of London, used two large Worthington pumps and five 100 horse-power boilers. This plant was capable of delivering to the mine, through a 20-inch flanged pipe, 5,500,000 gallons in twenty-four hours. The elevation of the point of discharge above the water-level at Swift Island, the point of supply, is 416 feet.

Russell Mine (p. 700).—Mr. Eames, the manager of this mine, writes (December, 1895), that "experiments have shown that the treatment by cyanide will cost, on a scale of 100 tons per day, 90 cents to $1 per ton, with an extraction of 85 to 90 per cent."

Parker Mine (p. 703).—Mr. Henry A. Judd, manager of this mine, writes (November, 1895); "Now down 131 feet in the Ross shaft and in diorite. At 125 feet, struck small quartz-vein, running $12 to $15 when first encountered; now somewhat fallen off in point of richness. The sulphurets are heavy, and carry some copper and a trace of silver. At 120 feet, started cross-cut east, to intersect a belt of quartz-veins. Have driven 45 feet, and expect to have to go from 10 to 15 feet further. Some surface-washing is carried on with water pumped from the shaft, and the returns are fairly satisfactory, being generally from 80 to 100 dwt. per month. A few weeks ago, we found a 3-ounce nugget."

Capps Mine (p. 711).—In the summer of 1895, Mr. Wilkes, the owner of the Capps mine, made, at his test-plant in Charlotte, a trial-run of 50 tons of Capps ore (sulphurets) from the old dumps; and the result of milling and chlorination was $27 per ton.

Brewer Mine (p. 762).—It is reported that at this mine, in the latter part of 1895, experiments with the cyanide-process, both without and with the aid of electrolysis, were carried on by Messrs. Motz and McNulty, and that the results were favorable.

Notes and Recollections Concerning the Mineral Resources of Northern Georgia and Western North Carolina.

BY WILLIAM P. BLAKE, MILL ROCK, NEW HAVEN, CONN.

(Atlanta Meeting, October, 1895.)

IN view of the present exposition at Atlanta, and the timely meeting of the Institute in that city, and the expressed desire of the Council for contributions to the knowledge of the mineral resources of the Southern States, I may be permitted to offer some desultory notes upon the subject, chiefly in the form

of recollections, which, though they may not possess at this date much direct technical value, especially in comparison with the results of more recent investigations, may serve to direct attention in a general and popular way to some of the mineral wealth known and partially developed as early as the middle of this century and the succeeding decade, and may also be useful as historical records, and as a means of measuring the scientific and industrial progress since achieved.

GOLD.

The gold-region of northern Georgia stretches diagonally through Lumpkin and White counties from the Carolinas into Alabama, following the eastern flank of the Blue Ridge, with Dahlonega as its approximate center. This gold-field, interesting alike to science, capital, and industry, for the extent, value, and variety of its deposits of the precious metal, has a special significance to the historian of gold-mining in that it gave to California, upon the discovery of gold at Sutter's Mill on the American river in 1848, an influx of gold-miners of skill and experience, who did much to introduce the most direct and effective methods of placer- and vein-mining. The influence of the Lumpkin county miners was felt everywhere on the ridges and in the canyons of the Sierra Nevada; and the rapid development of the dormant wealth of California, and the advancement of civilization on the Pacific coast, owe much to the hardy Georgians who had acquired their skill under the shadows of the Blue Ridge. But the gold placers of Georgia had in turn been opened up by the more enterprising miners from North Carolina. Even while the country was still occupied by the Cherokees, the outlying placers and creeks were secretly washed.

It may never be certainly known whether the credit of devising and introducing the hydraulic process of placer-mining should be given to the Lumpkin county miner or to the New York fireman; but the cradle, the sluice, the long-tom and the rocker were devices in use in North Carolina and Georgia before the exodus to California. But the once well-known Burke rocker, of Burke county, North Carolina, one of the most complete and successful forms of machinery for washing auriferous gravel, was not, to my knowledge, ever introduced in California.

The Lumpkin county gold-field, in Georgia, is the southwestward prolongation of the gold-field of Burke, McDowell and Rutherford counties of North Carolina. It may be followed across the western part of South Carolina by an agreeable drive through forests of black-jack, post-oaks and pine trees—a region opened, since the time of my travels in it, by a part of the Richmond and Danville railroad (now the Southern railway) system.

The rocks are all steeply plicated, compressed and eroded, and are probably of Huronian and Cambrian age. The itacolumite or flexible sandstone characteristic of the gold- and diamond-regions of Brazil, is found near the southern or eastern margin of the field and is described by Lieber, as is also the itabirite, developed chiefly in South Carolina.*

This western gold-field of the Carolinas and Georgia is distinct and separate from the gold-field, or belt, of Central North Carolina and Georgia. Being nearer to the Blue Ridge and the highlands, the valleys are deeper, the river erosion is greater, the gravels are coarser and the placer-deposits are more numerous and heavier than in the lower country. The western field is thus characterized by placer-deposits of gold rather than by veins, which are more pronounced and characteristic in the central field.

The principal streams descending from the Blue Ridge country and cutting across the gold-belt are the Catawba, Etowah, Chestatee, Yahoola and Cane creek. The alluvions of these streams and their branches are auriferous, and all of them have left hill-deposits and ancient channels, as in California, but on a much smaller scale. One notes also the absence of such vast outflows of lava as followed the ancient channels in California, entombing and preserving them, while forcing the streams to flow in other beds.

Brindletown Placer.—In North Carolina, the Brindletown placer is one of the best characterized examples of a hill-deposit. It was considerably worked by the hydraulic method, and by sluicing, about the year 1856.

Walton Branch Placer.—The Walton branch, near Rutherfordton, in Rutherford county, North Carolina, afforded one of

* Oscar M. Lieber, *Reports on the Geognostic Survey of South Carolina*, i., ii., iii., and iv., 1856–1860. Columbia, 1860.

the best examples of a stream deposit. It was especially interesting for the variety of rare minerals obtained with the gold, such as monazite, xenotime, zircon and corundum. It was worked energetically for some years.

The following extracts from an article on "Gold Mining by the Hydraulic Process in North Carolina and Georgia," by William H. Ellet, in the *Mining and Statistic Magazine*, vol. x., pp. 27–30, January, 1858, are given for the convenience of those to whom that magazine (long since discontinued) is not accessible:

"I avail myself of my earliest leisure to answer your inquiries in relation to the hydraulic gold-mining operations lately introduced by Dr. M. H. Vandyke, in some of the western counties of North Carolina. . . . . My observations on the hydraulic process were made during the month of April [probably 1857] at the Jamestown mine, in McDowell county, N. C. The water was there conveyed . . . . about 4 miles. The uniform descent was 4 inches to the hundred feet. . . . . The number of hose pipes employed was four. The mass of earth moved in nine working days was 20 feet in depth, 82 in length and 26 in breadth, being at the rate of 1184 cubic feet, or 966 bushels, per day for each hose. . . . . The labor employed . . . . was that of four men and two boys. . . . The yield in gold was $5.13 per day for each hose employed. . . . . During my visit to North Carolina, I examined with some care other localities which had been selected by Dr. Vandyke as the scenes of his future operations. These were in the adjacent counties of McDowell, Burke, and Rutherford."

The following is taken from the same periodical, vol. x., pp. 393, 394, May, 1858:

"The Wilkinson gold-mine in Burke county, N. C., is owned by Dr. Van Dyke and is worked by the hydraulic process. The water is brought . . . . by a canal or aqueduct for a distance of 15 miles. . . . . The water is not brought upon these mines at a very high head, only about 40 feet. There was only one pipe in operation at the time of my visit. The water passed through a 6-inch hose and a nozzle of 1½ inches. . . . . The average yield of the mine . . . . was about $5 a day to each hand. . . . . Obtaining a sample of the gold of this mine we passed over about 2 miles to the Bunker Hill mine, also in Burke county. This was formerly known as the Brindleton mine. It is owned and worked by Rev. Benjamin Hamilton. . . . . It is now worked by the hydraulic process. . . . . The amount of water is limited, sufficient only for about two pipes, which is brought in a small ditch for a distance of 4 or 5 miles. . . . . The Collins mine in Rutherford county is owned and worked by Dr. Van Dyke. The water is brought to this mine in a canal about 4 miles in length, at an elevation of 150 feet, and sufficient in amount for 20 pipes, and will command near 1000 acres of surface. . . . . Jamestown mine, McDowell county, N. C. [is] also worked by Dr. Van Dyke. The deposit workings embrace about 400 acres. The water is brought by a canal at an elevation of 70 feet, and is 5 miles in length. There is water enough here for 20 hose pipes."

The following is from a "Report upon the Gold Placers of

Lumpkin County, Georgia, and the Practicability of Working them by the Hydraulic Method, with water from the Chestatee River," by the writer of the present paper, in the *Mining and Statistic Magazine*, vol. x., pp. 457–476, June, 1858.

> "Desiring to see the results obtained [by Dr. M. H. Van Dyke] in North Carolina, and thus to be enabled to form a better judgment of the probable results in Georgia, I first visited the placers in Burke and McDowell counties where the [hydraulic] process is now in successful operation. . . . . The average yield, as shown by the results at several of the North Carolina placers, is about $6 a day to a pipe attended by two men, or by a man and a boy. At some of the placers the average is not less than $10 a day. . . . . At Brindletown, in the bed of a little brook which has a rapid descent, Mr. Hamilton has been washing very successfully with two pipes and five men and boys. . . . I am confident that the yield cannot be less than $20 a day, even among the former excavations where the gravel has been washed over more than once before."

Loud Deposit.—In Georgia, near Loudville, the celebrated Loud deposit is one of the most extensive and richest in gold in the Southern States. It is a fine example of an accumulation, in a basin-like area, of quartz-gravel broken from a vein, but well-rounded and water-worn. The quartz is all white and clean, and has the appearance of having been transported a considerable distance; but the gold is in a semi-crystalline condition and does not show marks of much attrition. The parent-vein cannot be far distant.

This deposit was worked and reworked for years, with fair success and in the ordinary way with long-toms and sluices. Mr. Asbury was one of the most successful miners in this deposit, and took out a large amount of gold. It is thought by those most familiar with the ground that if sufficient fall could be obtained for a ground-sluice, it would pay to rewash the entire deposit by the hydraulic method. A survey which I made to determine this point indicated that a tunnel driven through the ridge or rim of the basin would give the required depth and fall into the Chastain branch adjoining. It was supposed that the water for piping could be had at a sufficient elevation from Cane creek by means of a long ditch or canal.

As a general rule, it may be said that the high placer deposits of the region are deficient in the available fall necessary for successful hydraulic work, and that such deposits are not sufficiently extensive and rich to justify costly aqueducts and flumes on trestles. Several enterprises of this nature, such as

the Chestatee canal, the Yahoola, with a costly trestle, and the Cane Creek ditch, were not remunerative. Description in detail of these works, and of the placer ground about Dahlonega, may be found in the *Mining Magazine*, second series, i., pp. 77, 78, Nov., 1859; and pp. 360–366, March, 1860, and in descriptive pamphlet reports separately published in the years 1857 and 1858.

White County Placers.—The placers of White county have yielded a considerable amount of gold in large nuggets weighing several ounces. They are all much worn by stream-action, indicative of transportation from a distance. These nuggets compare, as to size and general appearance, with the coarser gold from the placers of California and with the coarse gold from Cabarrus county, North Carolina, where the largest mass yet found east of California was picked up in a brook. It weighed 28 pounds, and was used for a long time as a block to keep the door of a cabin open.*

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* See the *Mining Magazine*, vol. i., p. 513, November, 1853, and vol. iii., p. 162, August, 1854, also "A Sketch of the Discovery and History of the Reed Gold Mine, etc.," in *Historical Sketches of North Carolina*, by John H. Wheeler, vol. ii., p. 63, 1851, from which the following is quoted:

"The first piece of gold found at this mine was in the year 1799, by Conrad Reed, a boy of about 12 years old, a son of John Reed, the proprietor. . . . . The boy . . . . went to a small stream, called Meadow creek . . . . and . . . . while engaged along the bank . . . . saw a yellow substance shining in the water. He . . . . found it to be some kind of metal, and carried it home. . . . . The piece was about the size of a small smoothing-iron. . . . . Mr. Reed kept the piece for several years on his house-floor, to lay against the door, to keep it from shutting. In the year 1802 he went to market to Fayetteville, and carried the piece of metal with him, and on showing it to a jeweller, the jeweller immediately told him it was gold. [The jeweller paid him three dollars and fifty cents for the gold.] After returning home, Mr. Reed examined and found gold in the surface along the creek. He then associated Frederick Kisor, James Love and Martin Phifer with himself, and in the year 1803 they found a piece of gold in the branch that weighed 28 pounds. . . . . The veins of this mine were discovered in the year 1831."

The sketch from which the foregoing extract is taken was certified to in January, 1848, by George Barnhardt. The following are weights of different pieces of gold found at this mine:

	Pounds.		Pounds.
1803,	28	1824,	9½
1804,	9	"	8
"	7	1835,	13¼
"	3	"	4½
"	2	"	4
"	1¾	"	1
1824,	16	"	8

The "Pigeon-Roost Streak."—As an example of a peculiar class of lodes, beds or veins bearing gold, the "Pigeon-Roost Streak," so called, may be cited. It is not far from Dahlonega, and consists of micaceous schists or slates in which the presence of quartz-veins or beds is not at first apparent; but careful examination and actual working shows that the slates are permeated by quartz in thin layers or sheets, most of them not over ⅛ inch, or 2 to 3 millimeters, in thickness, interleaved with the micaceous layers. Occasionally, however, sheets from the quartz from ½ to 1 inch or more (say from 1 to 5 decimeters) are found, and all extending parallel with the stratification of the slates over a breadth or thickness of 10 to 20 feet. This forms a gold-bearing lode, or stratum, which has been worked for years with varying success. The general course of this bed, or assemblage of "knife-blade" veins, is marked by a rusted, decayed condition, due largely to the oxidation of pyrite, which, presumably, accompanies the quartz, as in ordinary lodes. It is in these decayed and softened slates, with the quartz and iron oxides, that the gold is mostly found. This gold is rather coarse, and close examination of the little sheets of quartz, after washing, will show gold adhering to and permeating this quartz.

Formerly, the miners of the district were content to work this "streak" by simple sluicing or cradling for the free gold. Later, a mill was erected upon the ground, and the soft slate and quartz were sluiced into and through the batteries, where they were crushed in the usual way. When first discovered, the surface was very rich, and, even after years of working, gold could be found after heavy rains in the run-ways of the drainage-water. The natural decomposition of the slates has, of course, greatly facilitated the extraction of the gold by simply crushing and washing. When the limit of the decay of the slates is passed, the mining will be much more costly and the gold will not be wholly free, but will be found—in part, at least,—locked in pyrites and possibly associated with tellurium.

Boly Fields Gold-Vein.—One of the most notable occurrences of gold and tellurium in close association was discovered on the banks of the Chestatee, in the compact hornblendic-gneiss or slate without any surface-decomposition. It consisted of a bunch or segregation without any well-defined extended out-

crop. It was exceedingly rich in coarse gold, both free and combined with tellurium. Thousands of dollars' worth of the metal were obtained in a few days by simply blasting it out of the rock, breaking it up with hammers and pounding the fragments in mortars.

This occurrence of so much free gold in a small space in slates, which, if not swept clean by the river-floods, would have been extensively decomposed and softened, is a good illustration of the manner in which placer-deposits of great value may be formed by river-action, without any indication of a great vein or fissure. The same observation may be applied to the Pigeon Roost streak or outcrop, by the erosion of which much gold would be accumulated in a placer without any large amount of visible quartz or evidence of a pre-existent vein.

The tellurium mineral of the locality was analyzed by Dr. Charles T. Jackson, of Boston, and referred to the species bornite.* Some criticisms of this result and additional information were given by Dr. F. A. Genth, of Philadelphia.† Much of the tellurium was thrown away by the miners. Several pounds could have been saved.

Auriferous Pyrites.—The writer is not aware of any attempts made in the south, prior to the war, to treat the sulphurets of gold-bearing veins by roasting and chlorination, which has since been successfully accomplished at the Haile gold-mine, S. C., by Mr. Thies,‡ and at other places. Formerly, when the limit of decomposition of the pyrites was reached at the water-level in the veins, and the gold was no longer free, the mine was

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* *Am. Jour. Sci.*, 2d series, xxvii., p. 366, May, 1859; and *Mining Magazine and Journal of Geology*, 2d series, i., p. 83, Nov., 1859.

The French mineralogist Beudant proposed the name "bornine" for a telluride of bismuth, and this name is given by Dana, in his synonymy, under tetradymite.

Dufrénoy also gave the name "bornine" to a seleniferous telluride of bismuth found in Brazil. This name is given by Dana, in his synonymy, under joseite.

Jackson found selenium in the Georgia mineral, and referred it to the Brazilian bornine (bornite) of Dufrénoy. He regarded it as distinct from tetradymite, which had previously been found in Virginia.

The controversy between Jackson and Genth has no reference to the use of bornite in its modern acceptation as equivalent to erubescite, etc.

† In the same periodical, 2d series, i., p. 358, March, 1860. Jackson's rejoinder to Genth appeared at p. 466 of the volume of the *Mining Magazine* last cited.

‡ See *Trans.*, xix., 601, and elsewhere.

abandoned (and, if possible, sold !), it being no longer profitable for working in the common way.

The practice of piling up the sulphides in heaps with salt, and permitting the mixture to remain exposed to the weather for years, was resorted to in some places in order to hasten and secure the decay or oxidation of the pyrite and the liberation of the gold for a subsequent amalgamation.

Holland's Process.—Dr. Holland, of Massachusetts, made some interesting efforts to oxidize pyrites, and set the gold free, by deflagration with nitrate of potash or nitrate of soda. This was in 1852 and 1853, at the mines near Charlotte, in North Carolina. He first obtained the pyrites as clean as possible by concentration in log-rockers, and then, after mixing with the nitrate, the mass was spread upon the floor of a reverberatory furnace and roasted at a low heat. This brought much fine gold to view in subsequent washing, but the same difficulties in amalgamation were met as always attend gold so set free.

Pyritic Smelting in Virginia.—At an earlier date (prior to 1847), efforts had been made at the old Vaucluse mine, in Virginia, under Commodore Stockton, one of the pioneers of gold-mining in the United States, to extract the gold from pyrites by fusion in a cupola-furnace. A salamander remained as a monument of the failure.

It is well known that gold-mining in the Southern States has not always been attended with brilliant financial success. Much of the disappointment and failure attending many of the enterprises has been due to extravagant expectations and over-capitalization, for the purpose, generally, of floating shares upon the public. Many "prospects" which would have made handsome returns upon the money necessary to work them have failed to reimburse those who invested their money at perhaps ten or twenty times the value of the ground. Investors and promoters are probably equally at fault in permitting their cupidity to overbalance sound business judgment and expert advice in such matters.

Silver.

Silver Hill Mine.—Probably the only mine in the United States which could, with propriety, be called a silver-mine prior to the discovery of the Comstock lode, was worked in Davidson

county, North Carolina. It was a mine of argentiferous lead, and was notable amongst mineralogists for the size and beauty of the crystals of carbonate of lead and other crystalline products of the alteration of the galenite, found in the upper portions of the lode.

The ores of this locality were represented in the mineral collection at the Crystal Palace, in New York, in 1853. See the *Mining Magazine*, ii., p. 605, June, 1854. A description of the mine, with drawings, was published in the *Mining Magazine and Journal of Geology*, 2d series, vol. i., pp. 428–435, April–July, 1860. See also same vol., p. 368.

An earlier description is found in the first series of the *Mining Magazine*, vol. i., pp. 360–370, October, 1853, under the name of the Washington Mine, from which the following is quoted:

> "The ore which this company worked has yielded from 200 ounces to 300 ounces of silver to the ton of lead. It contains, on the average, 8 per cent. of lead. . . . . The silver has been worth $1.80 the ounce, because it was alloyed with a large portion of gold."

The same article gives a description of the methods of extracting the silver by smelting, refining, etc., with illustrations of furnaces, but no mine-drawings.

Silver-Ore in Tennessee.—No satisfactory explanation has ever been given of the reported occurrence of a mass of silver sulphide* in eastern Tennessee. The source of the mass has not been found.

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* *Mining Magazine*, viii., p. 238, March, 1857, from which the following is quoted:

"Independent of the trace of silver-ore to be found on analysis in the lead-ores of Tennessee, the late Dr. Troost describes in his Fifth Report, for 1834, two specimens of the sulphuret of silver found by him in the waters of the Cumberland mountains. . . . . We will give his own account:

'I had the good fortune, during my last excursion, to make a discovery which may, eventually, be of great importance. Stopping for the night at the house of Captain Eastland, on Clifty creek, ... he handed me some small fragments of ore. . . . . Next morning I left that place, and, passing through Sparta, I descended to the Calf-killer creek to water my horse; my attention was there attracted by something uncommon among the gravel; I dismounted and took up the substance which had drawn my attention. . . . . When at home, I examined both these ores, and found that the fragment of Captain Eastland, as well as that found by me on the Calf-killer, was sulphuret of silver. . . . . I am at a loss to make any conjectures as to the locality of this ore. . . . . I do not know from whence

Copper.

The Ducktown, Tennessee, Deposits.—These were brought prominently into notice in 1852, and capital from Providence, R. I., and New York City was enlisted in their development. In the spring of 1853 they were visited by me in company with Prof. J. D. Whitney, and their nature and the formation of the secondary deposit of rich black ore was then first diagnosed and described.*

It was evident that the large bed of low-grade pyrrhotite, with its included grains and bunches of yellow copper pyrites, had suffered decomposition from the surface down to the water-level of the region, with the formation of copper-sulphate and hydrous iron oxide or gossan. The gossan remained in place, while the copper sulphate percolated downwards along the bed until it came into contact with the surface of the decomposing sulphide, where it was reduced to a black, soft mixture of oxide and sulphide sufficiently rich in copper to be profitably smelted. It there precipitated and accumulated, and formed a layer on the top of the unchanged pyrite, varying from a few inches to several feet in thickness.

The subsequent operations upon these beds and ores have been described by Mr. Henrich in a recent paper before this Institute.†

The duality and parallelism of some of the outcrops of these beds suggest their interstratified or imbedded origin, and their subsequent plication with the inclosing rocks, the deposits being synchronous in origin, and not veins occupying fissures.

Copper Ore in Gold-bearing Veins.—Many of the auriferous quartz-veins, both of Georgia and of the Carolinas, carry a large percentage of sulphides, of which yellow copper-ore, chalcopyrite, forms a large part. This is true especially of the ores of Conrad Hill, North Carolina, and of Canton, Georgia. In some places this mineral is associated with siderite—carbonate of iron. At Canton occurs the altered mineral, which was named "cantonite" by Prof. Shepard. The ore at the locali-

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Capt. E. obtained his specimen—as he told me it was about fifteen miles from his residence.' "

* *Metallic Wealth of the United States*, J. D. Whitney, 1854.

† "Ducktown Ore-Deposits," etc.; p. 173 of the present volume.

ties named is not, however, sufficient in quantity to justify independent smelting-operations.

ZINC.

No important occurrences of the ores of zinc are known in the Carolinas or Georgia; but in Tennessee, at Bald Hill, near Knoxville, there are considerable deposits of smithsonite—"bone"—in the limestones. These are, doubtless, the oxidized upper croppings of portions of blende—"black-jack."

A description of these deposits may be found in the *Mining Magazine and Journal of Geology*, 2d series, i., pp. 419–427, April–July, 1860.

IRON AND COAL.

With regard to the extensive and varied resources in iron-ores and coal of Alabama, North Carolina and eastern Tennessee, it is sufficient to refer to the extended literature of the subject to be found in our *Transactions*. The same remark may be made as to manganese.

The iron-ores of the Cranberry region and of eastern Tennessee generally, were exhibited on a most liberal scale at the International Exhibition, in Philadelphia, in 1876, through the exertions of General B. F. Wilder. It was an impressive display, and aided greatly in arousing increased interest in the immense resources in iron of that section. At the close of the Exposition, a visit was made to Cranberry and its vicinity, by Lowthian Bell and John Lancaster, of England, together with the writer, as the guests of General Wilder.

MINERALS OF SPECIAL INTEREST.

The early knowledge of many of the minerals of northern Georgia and of the western counties of North Carolina, was much promoted by the enthusiastic devotion of two men—Dr. W. F. Stephenson, of Georgia, and Senator Thomas L. Clingman, of North Carolina. Both were indefatigable collectors, and brought many remote localities of peculiar minerals to the notice of scientific mineralogists.

The variety of mineral species, more especially interesting to science, yet daily finding new technical and industrial uses, is very great. Space will not permit the giving of a complete list; but a few of the more important minerals will be briefly

noticed. The literature of the subject has largely increased since 1863; and the mineral cabinets of the world have been enriched by specimens from the Carolinas and Georgia.

Rutile.—The magnificent specimens of crystallized rutile which adorn so many collections, come from Graves mountain, in Washington county, Georgia. They occur loose in the soil, having been weathered out of a vein or bed of radiated talc and oxide of iron, and vary in size from a pin's head to prisms six inches in diameter. The surfaces of the crystals freshly broken from the envelope of rusty iron oxide, are remarkably splendent and beautiful, with a mirror-like polish and very perfect angles and planes.

Active work at the locality began in 1857, when, visiting it in company with Dr. Stephenson, Professor Charles U. Shepard arrived, and we unitedly made arrangements for the acquisition of the property, and its systematic working. It subsequently became the property of Professor Shepard.

Rutile is also found in good crystals in North Carolina.

Ilmenite.—This titaniferous mineral, the washingtonite or Shepard, is a common associate of the quartz-veins of the gold-region of Georgia, and its fragments are abundant in the black sands and residues of gold-washing, where they occur together with magnetite and hematite, and the other heavy minerals.

Cassiterite.—A laborious search through the heavy sands and residues of gold-washing in the placers of Georgia failed to show the presence of tin, except in one instance, where some grains of wood-tin were discovered in black sand.

In Alabama, however, tin-ore has been shown to exist in place, and the locality has been partially opened and prospected.

It has also been found in Virginia.

Lazulite.—This occurs in a white granular quartzite at Graves mountain, Georgia, and also in North Carolina. The Georgia locality affords the finest crystallizations, much sought for and prized by mineralogists.

Talc.—A snow-white talc, of a peculiar softness and texture, occurs, massive and in abundance, in the Nantahela valley, North Carolina. It has become useful, and is highly valued as an emollient.

Mica.—The mountains of the western part of North Caro-

lina have supplied large quantities of muscovite mica of excellent quality to the manufacturers of stoves. The industry has assumed large proportions, and gives employment to many households during the winter months.

Zircon.—North Carolina affords this mineral in large quantity. From a mineralogical curiosity, brought first to notice by General Clingman and Professor Shepard, it has become industrially important, and is exported in considerable amount.

It occurs also in the midst of the iron-ore of the continuations of the Cranberry vein or bed.*

In western North Carolina, especially at the Walton branch, worked for gold, transparent white crystals, and also some slightly colored ones, could be found in the sluices when the gold was cleaned up. Their high lustre and index of refraction, with the curved surfaces of fracture, make them resemble diamonds.

In some places in North Carolina, after a shower of rain, the roadsides are sheeted with white zircon sand, composed of minute but well-formed crystals, with the adamantine lustre of diamonds. This beautiful sand is spread over the surface in the water-ways precisely as black sand is ordinarily so found.

Corundum.—The abundance of corundum in North Carolina, was early suggested by the discovery of a large bowlder of the blue, cleavable variety, which Senator Thomas L. Clingman brought to the notice of Professor Shepard. This mineral species has since been found in a variety of forms in the beds of most of the streams where gold has been washed out, and has become industrially important, not only as an abrasive but for the production of metallic aluminum in the electric furnace. The late C. W. Jenks, a member of the Institute, gave special attention to the development of one of the most promising localities of this species, and secured many specimens of different colors and a fair degree of transparency, valuable as gems.† The valuable work on gems by Mr. George F. Kunz may also be consulted in regard to the occurences of corundum, rubies, and other gems in the Southern States.

Hiddenite.—This transparent green variety of spodumene is

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* See my note on the subject, *Trans.* vii., 76.

† See Dr. Raymond's paper on the Jenks Corundum mine, *Trans.*, vii., 83.

entitled by its beauty and rarity to rank as a gem, and to the distinctive name it bears. It was obtained about 1876 from Alexander county, North Carolina, and has been described by J. Lawrence Smith and Genth* and by George F. Kunz.†

Monazite.—This rare mineral occurs in the auriferous gravels of the Walton branch, in Rutherford county, North Carolina, as already mentioned, and in other places, from which it has been obtained in quantity to meet an industrial demand. When discovered there in 1857, it was regarded as only interesting to mineralogical science. It occurs in small tabular crystals of a honey-yellow or brownish-yellow color, from one-eighth to one-quarter of an inch in length.‡

Building Stone.

Before the war but few quarries had been opened in the mountain region, but efforts had been made to get the beautiful white statuary marble of Valley river into notice.

Sub-Aerial Decay of Crystalline Rocks.

The phenomena of deep decay or decomposition of granite and gneissic rocks in place are shown in a most striking and interesting way in the cuttings along most of the railways of the South. The firmest granite has lost its hardness and can be cut with shovels like indurated clay, while the quartz alone retains its original hardness and shows by its form, if in veins, the original structure of the rock. These phenomena excited the interest of Sir Charles Lyell upon his first visit to the United States, and have since been the subject of discussion by many geologists. The practical bearing of this decomposed softened condition will be appreciated by placer gold-miners, who will recognize the large amount of vein-stuff loosened and set free by the alteration, and left in a condition for rapid erosion and cutting away by water, so as to form placer-deposits. A similar decay of the gold-bearing rocks is found in California. The phenomena are novel to those unaccustomed to unglaciated regions.

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* *Am. J. of Sc.*, 3d series, xxi., 128 (1881) and xxiii., 68 (1882).

† *Gems and Precious Stones*, New York, 1890.

‡ See the paper of Mr. Mezger, page 822 of the present volume.

The extent of the decay of granite ledges is probably the best index that can be had of the capacity of the rock for resistance to weathering. The presence of an abundance of iron pyrites in either granite, gneiss or slate, is a great factor in their rapid decomposition.

The Phosphates and Marls of Alabama.

BY EUGENE A. SMITH, STATE GEOLOGIST, UNIVERSITY, ALA.

(Atlanta Meeting, October, 1895.)

Geological Relations.

In his second report upon the Geology of Alabama, Prof. M. Tuomey calls attention to a rock occurring near Florence, in the Tennessee valley, the composition of which is as follows:

	Per cent.
Carbonate of lime,	16.41
Phosphate of lime,	14.19
Peroxide of iron,	.36
Quartz and other insoluble matters,	68.72
	99.68

This rock was referred by Prof. Tuomey to the Carboniferous, because he found the Carboniferous limestones always resting conformably upon it, whilst both rest uncomformably upon the underlying Silurian or Devonian rocks.

The recent discovery in Tennessee of phosphate-rock of this age (or approximately so) has stimulated the search in Alabama for similar rich material; but our search has been, up to this time, unrewarded by the finding in quantity of anything of better quality than the rock whose analysis is given above. It is, however, in high degree probable that the rich phosphate-rock of Tennessee will at some time be found to have its counterpart in Alabama.

Materials of several kinds, containing a notable quantity of phosphoric acid, have been found in Alabama in both the Cretaceous and the Tertiary formations, and for our present purpose they may be most conveniently considered in their geological relations.

The strata of these two formations occupy the surface and constitute the mass of the coastal plain of Alabama, which embraces about three-fifths of the area of the State. These strata have a gentle slope or dip of 25 to 40 feet to the mile towards the Gulf of Mexico, and outcrop across the State in approximately parallel belts, the older beds further towards the north, or from the gulf border, and the newer further towards the south, in the order of their relative ages. As another consequence of this dip of the strata, each one of these beds, while appearing at the surface only in a comparatively narrow belt, passes beneath the newer beds, and may be found below the surface at depths constantly increasing as we go southward.

The geographical distribution of the surface-outcrops of the several divisions of the Cretaceous and Tertiary may be seen by reference to the geological map of the State recently issued, and a few words descriptive of the component strata of the several divisions of these formations will be necessary to the proper understanding of much that follows.

Cretaceous.

Tuscaloosa.—The oldest of the beds which have been referred to the Cretaceous constitute what we have named the Tuscaloosa formation. This consists of about 1000 feet of clays and sands—more clayey below and more sandy above. Some of the clays of this formation contain leaf impressions, from which it has been possible to determine the geological horizon as approximately that of the Raritan clays of New Jersey. In the lower part of the formation there are also beds of massive clay which bid fair to come into extensive use in the manufacture of fire-brick and of the various kinds of fine earthenware. In the same formation are also some beds of yellow ocher which have been worked on a commercial scale.

A very good quality of brown iron-ore is likewise found in many places in the territory of the Tuscaloosa formation. It has been used in a furnace at Vernon, in Lamar county.

Eutaw.—Above the Tuscaloosa formation come some 300 feet of strata, chiefly sandy, to which the name of Eutaw has been given. These have not, as yet, yielded anything of economic value.

Rotten Limestone.—Next above the Eutaw follow the calcare-

ous beds of the Rotten Limestone or Selma Chalk, 1000 feet in thickness and consisting of indurated calcareous clays or clayey limestones, in many parts filled with the microscopic shells of *foraminifera*, and therefore of the nature of chalk. At the base of the Rotten Limestone we find one of the beds of phosphatic green-sand and phosphatic nodules, to which attention will be directed later.

Ripley.—Above the Rotten Limestone are 250 feet of Ripley strata, as they have been called—calcareous below, sandy above, and holding in the lower part another important bed of phosphatic materials, also to be again referred to more in detail.

TERTIARY.

Eocene.

Clayton.—At the base of our Tertiary we find a valuable bed of calcareous material, running in thickness from 25 or 30 feet in western Alabama to over 200 feet along the Chattahoochee river. East of the Alabama river the Clayton holds beds of limestone of sufficient magnitude to give rise to caves and lime-sinks and big springs.

Lignitic.—About 1000 feet of strata immediately overlying the Clayton constitute what we have named the Lignitic division, from the occurrence of beds of this material among its strata. The Lignitic division is also marked by the occurrence of several beds of marine shells, which serve to designate its several subdivisions. Of these the *Nanafalia* and the *Wood's Bluff* or *Bashi* marls seem destined to be of importance to our agriculture in the future.

Claiborne.—Next above the Lignitic follows the Claiborne group, with its two subdivisions, the *Buhrstone* below and the *Claiborne* proper above. The first of these, 300 feet in thickness, is made up of strata chiefly siliceous, among them some beds of clay with the siliceous remains of *radiolaria*, marine diatoms, and *foraminifera*, constituting a sort of "Tripoli." The second division, 150 feet thick, the Claiborne proper, is remarkable chiefly for the great abundance and admirable state of preservation of the shells which occur in its upper part. These beds might be advantageously used where a merely calcareous marl was desired.

St. Stephens or White Limestone.—Above the Claiborne, and

forming the uppermost of the Eocene deposits of this State, is the White Limestone, 200 or 300 feet in thickness, in which we have found some phosphatic beds which may hereafter find some use. A part of the limestone forms a building-material good for many purposes, and a part of it yields, on burning, an excellent quality of lime.

Miocene.

Grand Gulf.—The strata of this formation, essentially sands and clays, often indurated into firm resistant rocks, and of an estimated thickness of about 500 feet, or possibly more, contain nothing as yet observed of economic value.

Pascagoula.—The uppermost strata of the Miocene, consisting of greenish and bluish clays, with numerous estuarine fossils, as well as marine diatoms and *foraminifera*, have been penetrated for many feet in artesian borings at Mobile and Pascagoula, but their appearance at the surface has not been made certainly clear, except at a few points.

With this summary, we may go on to the discussion of the phosphatic materials occurring at the various horizons enumerated, taking them up in the order of their ages, the oldest first.

Phosphates of the Cretaceous Formation.

I.—*The Hamburg Bed.*

This was the first discovered of our phosphatic strata; and in many respects it is the most important. Its geological position and the character of the strata may best be seen in the following section at Hamburg, Perry county:

1. Base of the Rotten Limestone. The beds here included contain very little phosphate of lime.

2. Green-sand, averaging about 5 feet in thickness, and strongly impregnated with phosphoric acid. The lowermost strata of this green-sand contain a few phosphatic nodules, which, however, become more abundant in the next succeeding stratum.

3. Sandy calcareous stratum (matrix of the nodules), 5 to 6 feet in thickness. Where this bed outcrops in the fields, the

surface of the ground is covered with the phosphatic nodules, which are concretionary masses of nearly pure phosphate of lime, of exceedingly irregular shape, and varying in size from that of pebbles no larger than a pea, to pieces two or more inches in diameter. The nodules also vary in color from light gray to dark brown; and they may easily be recognized by the peculiar odor which they emit when rubbed together or broken. This odor is described by some as "fishy," by others as "bituminous" or "naphthous," as it resembles in some degree the odor of crude petroleum. Along with these phosphatic nodules occur also great numbers of casts or moulds of fossils, usually somewhat worn and badly preserved, and consisting of fragments of *ammonites*, *baculites*, *nautili* and other well-known Cretaceous forms. These casts have themselves, in most cases, been more or less completely phosphatized, and in some instances have almost entirely lost their original shape and structure, and are then difficult to distinguish from the nodules. Besides these, there are great numbers of the teeth of sharks and bones of saurians.

The nodules are found, as above stated, in small numbers in the lower layers of the green-sand, but more abundantly in the next underlying stratum, mainly through about 2 to 2½ feet of its thickness.

From Hamburg this bed has been traced westward to the Mississippi line, and eastward nearly to Georgia, but with varying content of phosphatic materials. These materials, classified according to the percentage of phosphoric acid, are:

1. *The Nodules and Shell-Casts.*—Numerous analyses of these nodules and casts show that they contain from 20 to 30 per cent. of phosphoric acid, and are therefore very similar in composition, as they are also in appearance, to the nodules that make up the bed of phosphate-rock occurring in South Carolina. In the latter State, however, the nodules are aggregated into a tolerably compact mass, several inches in thickness, while with us they occur disseminated through 5 feet or more of strata, and, so far as we now know, too sparingly to be of commercial value. About Hamburg, however, and in some other places, the nodules lie loose upon the surface in large quantity, probably representing the *débris* from the wearing away of the containing rocks during many ages. These might easily be

collected with little expense, and in value would be about equal, weight for weight, to the average of the Charleston phosphate-rock.

2. *The Green-Sand.*—This bed, about 5 feet in thickness, contains a notable amount of phosphoric acid, varying from 2 to 5 per cent., with an average, shown in eight analyses of average material from about Hamburg, of 4.6 per cent. of phosphoric acid.

3. *The Matrix of the Nodules.*—Analyses of eleven typical specimens from about Hamburg give an average of 2.5 per cent. of phosphoric acid.

Except in Hamburg, the green-sand and the matrix of the nodules have not been separately examined, but specimens from the entire phosphatic stratum have been lumped together. These specimens, collected from all parts of the belt, extending from Eutaw across to Elmore county, show, as the result of forty or more analyses, an average content of 3.33 per cent. of phosphoric acid, which may fairly be taken as representing the composition of the entire bed in this respect.

II.—*The Coatopa Bed.*

In geological position, this bed immediately overlies, as the Hamburg bed immediately underlies, the Rotten Limestone. It comes to the surface as a border along the southern edge of the Rotten Limestone from the Mississippi line across into Macon county, Ala., beyond which it has not yet been followed. As at Hamburg, the materials are:

1. Phosphate nodules and shell-casts, similar in appearance and in content of phosphoric acid to those above described.

2. A green-sand marl, with an average of 1.5 per cent. ot phosphoric acid, and about 30 per cent. of carbonate of lime in a fine earthy powder. The thickness of this bed is about 5 feet, and it has been seen at intervals from Coatopa in Sumter county to near Bragg's store in Lowndes county.

III.—*The Linden, Prairie Bluff, Snow Hill Bed.*

This bed occurs near the summit of the Ripley division of the Cretaceous, and consequently near the summit of the Cretaceous itself. Its materials are entirely similar to those men-

tioned above, viz., phosphatic shell-casts and nodules of high grade (20 to 30 per cent. of phosphoric acid), and phosphatic green-sand marl, with 1.5 per cent. of phosphoric acid and from 20 to 30 of soft pulverulent carbonate of lime. This has been used with very good results by Mr. W. S. Purifoy, of Furman, in Wilcox county. In addition to the above, a sandy limestone with some 8 to 12 per cent. of phosphoric acid is met with in many localities in the lower part of Dallas, and adjoining parts of Wilcox, county.

In several places in the lower part of Marengo county, notably near Dayton and near Nixonville, there occur tolerably compact beds of shell-casts, containing from 20 to 25 per cent. of phosphoric acid. These beds hold out, perhaps, more promise of commercial value than any other phosphatic deposits of the State.

While the phosphatic concretions seem to be more abundant in the three horizons named, they are yet by no means wanting in other strata of the Cretaceous, and especially in the Rotten Limestone, in which they occur sparingly throughout its entire thickness.

Phosphates of the Tertiary Formation.

We have observed the occurrence of phosphatic materials in at least four different horizons of our Tertiary, viz., in the *Black Bluff* and *Nanafalia* sections of the Lignitic; in the *Claiborne*, at Ozark, in Dale county, and in the *St. Stephens White Limestone.*

1. *Black Bluff or Sucarnochee.*—In the eastern part of Wilcox county the prairie clays derived from the disintegration of the strata of this section are highly fertile, and although no analysis of any of these soils has been made, it is probable that they are exceptionally strong in phosphoric acid; for they are in places filled with irregularly-shaped concretions, or coprolites, which are chiefly phosphate of lime, containing some 30 per cent. or more of phosphoric acid. Some of the sandy limestones, also, of this section (or just above it) are phosphatic, containing up to nearly 4 per cent. of phosphoric acid.

2. *The Nanafalia.*—The important bed in this section of the Lignitic is a deposit of shells of a small oyster, *gryphæa thirsa.* In some localities we find the bed 15 to 20 feet in thickness,

consisting of an almost compact mass of these shells packed in green-sand. At intervals through this bed are indurated ledges, which very generally contain a notable amount of phosphoric acid—in one instance reaching 6.7 per cent.

3. *The Claiborne.*—At this horizon, in Dale county, there is a shell-marl with green-sand, 5 feet thick, carrying a very considerable percentage of phosphoric acid. To judge from the large amount of carbonate of lime in pulverulent form which it holds, this marl would be an exceptionally good fertilizer.

4. *The White Limestone.*—In the lowermost 50 or 60 feet of this formation we have noted the occurrence both of phosphatic concretions or nodules and of phosphatic marls. The mingling of these marls by natural agencies with the soils lends to the latter very great fertility, as may be seen from the vegetable growth upon them.

So far as we know, there have been made, as yet, no practical tests of these Tertiary phosphates.

Calcareous Marls, not Phosphatic, of the Cretaceous and Tertiary Formations of Alabama.

Cretaceous Marls.

In the upper part of the Eutaw formation, throughout the Rotten Limestone and in many parts of the overlying Ripley there are beds of calcareous materials which might be profitably used upon the fields, where the cost of applying them is not too great. Most of these marls have been spoken of in connection with the phosphatic marls with which they are commonly associated, and there appears to be little need of adding to what has already been said.

Tertiary Marls.

The principal horizons at which these occur are:

1. *In the Nanafalia* section of the Lignitic, already mentioned in connection with phosphatic marls. This marl-bed appears in the Bluffs of the Tombigbee river at Nanafalia landing, and thence down the river for several miles to Gay's landing, in Marengo county; on the Alabama river it appears at Black's and Gullette's Bluffs and for a short distance above the last-named place. Between the two rivers the bed crosses Marengo and Wilcox counties, and thence it may be traced across the

country to the Chattahoochee river at Fort Gaines. As has been said already, this bed is made up of small oyster-shells packed in green-sand, and it is one of the most widely distributed and most uniform in composition of the marl-beds of the State. Its fertilizing action is everywhere demonstrated where it has come to be mixed with the soil by natural agencies.

2. *The Wood's Bluff or Bashi marl*, like the preceding, is very widely distributed in the State; its outcrop has been followed from the Mississippi line across to the Chattahoochee river. It contains a large percentage of green-sand, as well as of decomposed shells, which furnish carbonate of lime in very available form, almost pulverulent. Its fertilizing properties are amply demonstrated at hundreds of places by the vigorous growth of plants along its outcrop.

3. *In the Claiborne section* of the Tertiary there is no lack of good shell-marls, since the uppermost 150 feet are composed almost exclusively of calcareous beds, many of which contain the lime in such proportion and in such form as to render them suitable for use.

4. *The White Limestone.*—As its name implies, this is a calcareous formation. In its lower part the composition is that of an indurated marl similar to much of the Rotten Limestone of the Cretaceous; and in its disintegration into soil this resemblance to the Rotten Limestone is maintained. The upper part of the formation contains a large amount of soft chalky limestone that could easily be pulverized and rendered fit for application to the fields, which it would undoubtedly, in almost every case, greatly benefit.

Economic Relations of the Alabama Phosphates.

The agricultural value of the several varieties of marls existing in Alabama depends, first, upon their content of phosphoric acid, lime or potash—in other words, upon their fertilizing power; and, second, upon their abundance and the cost of application to the soil, *i.e.*, upon their availability.

From the many analyses made of these materials it will be seen, as above shown, that, in addition to the simply calcareous marls, we may distinguish three grades of phosphatic materials, viz.:

a. Phosphatic concretions, nodules and phosphatized shell-casts,

containing from 20 to 30 per cent. of phosphoric acid, and therefore of the same nature as the high-grade phosphates of South Carolina.

b. Phosphatic green-sands, with very little carbonate of lime, but with an average of 4.5 per cent. of phosphoric acid. With these might also be classed the strongly phosphatic siliceous limestones which have their position in the upper measures of the Ripley formation.

c. The green-sands containing only about 1.5 *to* 2 *per cent. of phosphoric acid*, but, on the other hand, from 15 to 20 per cent. of carbonate of lime in a loose, pulverulent form.

As to value, there can be no question about the first class of high-grade phosphates, except that of quantity. In this respect, unfortunately, Alabama seems to fall behind; for we have not yet observed, except in the cases mentioned in Marengo county, any very considerable quantity of these phosphates in the State.

With regard to the second and third classes, there may be difference of opinion about the fertilizing action of the small percentages of phosphate which they carry. We have, however, in New Jersey, an actual demonstration of the value of phosphatic marls, which have been for many years very generally used upon the soils of that State. The experience of the New Jersey farmers has shown that their marls are valuable, in proportion, first of all to the percentage of phosphoric acid which they contain, and, second, to the proportion of carbonate of lime in soft, pulverulent form.

Now the average marl from the Lower bed (New Jersey) contains 1.14 per cent. of phosphoric acid, which is not high, but, in addition, the marl contains 10 to 20 per cent. of carbonate of lime in fine powder.

The average of five analyses, given as showing the composition of the green marls of the Middle bed, shows about 2 per cent. of phosphoric acid. Many hundreds of tons of this marl are shipped every day over the railroads to the different parts of that State.

The average of four analyses given as representing the composition of the marl of the Upper bed shows about 3 per cent. of phosphoric acid. This marl is also shipped by rail to all parts of the State.

The above-quoted averages are considered by the Department of Agriculture of New Jersey as fairly representative of the character of the three grades of marl in general use in that State.

The reports of the State Geologist and of the Department of Agriculture of New Jersey, show to demonstration that the use of these marls has improved the originally barren sands of New Jersey until they are worth more per acre than the agricultural lands of any other State in the Union.

If we compare the phosphatic marls of Alabama with those of New Jersey, we shall find that they are in many respects quite similar.

Thus, the average of eight analyses of the typical Hamburg green-sand gives us 4.6 per cent. of phosphoric acid.

Analyses of eight typical specimens of the matrix of the nodules at Hamburg give an average of 2.5 per cent. of phosphoric acid.

The analyses of forty specimens of the mixed green-sand and matrix from many localities away from Hamburg show an average of 3.33 per cent. of phosphoric acid.

The average of eight analyses of the green-sand from the upper or Ripley horizon shows 1.44 per cent. of phosphoric acid and over 30 per cent. of soft, pulverulent carbonate of lime.

Without carrying the comparison any further, it is easily seen that the phosphatic marls of Alabama, from two distinct horizons, have very nearly the same composition, as regards phosphate and carbonate of lime, as those which have wrought such a change in the agricultural practice of New Jersey; and it is reasonable to suppose that, when our people shall come to use these natural fertilizers as generally as do the farmers of New Jersey, a similar great change will be effected in our agriculture.

At the present time our citizens seem to be more concerned in the production of something to sell, than of something with which to enrich their lands.

If we compare the two States of New Jersey and South Carolina, one of which has only low-grade phosphates suitable for use at home, but not rich enough for export, while the other possesses high-grade phosphates which are shipped to all parts

of the world, we can hardly fail to see that the advantage to the State at large is in favor of New Jersey, with its low-grade marls. In South Carolina the phosphates are either exported or manufactured into high-grade fertilizers, from which, in all probability, the soils of that State derive no more benefit than do those of other States. The marls of New Jersey, on the contrary, are used almost exclusively at home, and the result, as already remarked, has been to make the lands of New Jersey worth more per acre for agricultural purposes than those of any other State.

If Alabama had an abundance of high-grade phosphates only, it is probable that our experience would not be different from that of the people of South Carolina. Our soils would, as a rule, be no better for it, since the greater part would undoubtedly be exported to enrich the soils of some other country.

If our marls and phosphates be rationally used by our people, there can be no doubt that the enhancement in value of our lands and the increase in the crops due to such use, will represent a larger amount of capital than would the trade in exported rock. And even if the sums to be realized by exportation were greater, the money would be in fewer hands, and would therefore accomplish less general good.

The Monazite Districts of North and South Carolina.

BY C. A. MEZGER, M.E., SHELBY, N. C.

(Atlanta Meeting, October, 1895.)

THESE districts are limited approximately on the north by the railroad from Salisbury to Ashville, and comprises the counties of Alexander, Catawba, Burke, Rutherford, McDowell, Cleveland, Lincoln, Polk and Henderson in North Carolina, and Greenville, Spartanburg, Union and York in South Carolina. The occurrence of monazite at Amelia Court House, in Virginia, belongs evidently to the same geological system, as its distance from the Alleghenies corresponds with that of the above-mentioned districts, and all the stratification runs parallel with those mountains.

I do not hesitate to give to the whole formation the name of eye-gneiss (*Augengneiss*), and to pronounce it identical with the eye-gneiss of Rio de Janeiro, in Brazil, concerning which a somewhat voluminous literature exists. It is a very curious formation; its characteristics consist in layers of gneiss, as a rule rich in mica, bent around lenses of granite, mostly of coarse grain. The proportion between the thickness and the length of the lenses varies between 1 to 1 and 1 to 20 or more; they range in diameter from 1 foot to 20, or more; perhaps even considerably more, as will be seen further on.

The place where this formation is most beautifully shown is a railroad cut about 100 yards east of the station-house of Claremont, on the Salisbury and Ashville line. Here the proportion of thickness to length of the lenses is about 1 to 1½, the diameter being about 1 foot, and the regularity very great.

The building-stone at Morgantown, of which many stone steps are made, shows the structure very well. It consists of lenses, the proportion of which is about 1 to 10, the thickness being only a few inches.

For the first mile from Morgantown the banks along the road leading from that place to Shelby likewise show the structure well, though with some irregularities.

Near Cleveland Springs, 3 miles from Shelby, N. C., the lenses have a size of from 3 to 6 feet, but here the formation is utterly unrecognizable, as the lenses are nowhere visible, and the surface of the rock is always the gneiss, showing, however, the bending and the waves characteristic for the formation.

In the mountains on the line between Cleveland and Burke counties, we find the lenses, consisting of a perfectly disintegrated fine-grained granite, containing some 0.2 to 1.0 per cent. of monazite. The lenses attain several hundred feet in extension.

The Rio de Janeiro formation, already mentioned, shows this difference: that the gneiss-layers often appear (on the surface of fence-posts, columns, steps, etc.) as fine as pencil-lines, only one or two millimeters thick. I have never found them so thin here. In Claremont, they are about one inch thick; in Cleveland Springs, and all around Shelby, nearly one foot, often more; and in the mountains, 20 feet and more.

The monazite, mainly a phosphate of cerium, lanthanum,

didymium, erbium, thorium, yttrium, terbium, etc., often containing silicates, perhaps of the same oxides, and often developing argon, if treated with sulphuric acid, is contained in the granitic parts of the eye-gneiss, and has been found by me only in partly or wholly decomposed orthoclase, never in the fresh unaltered mineral. There can be no doubt that that monazite which now forms the object of a quite important industry has undergone a concentration in the creeks to such a percentage as permits profitable work. I have found, however, many places where the rock will pay for crushing and concentrating; the more so, as, according to my experiments in these cases, the monazite appears to be considerably purer than the material obtained from the creek-sand,which is mostly mixed with garnet (pyrope), chromic iron, zircon, a low-grade thorite, diamonds (rarely, on account ot their lower specific gravity, which causes them to go off with the tailings), and other minerals of the gravity of monazite. Gold also occurs quite frequently.

Henderson county produces auerlite, zircon, and a large number of other rare and most interesting minerals, containing the oxides of the rare elements mentioned above. Curiously enough, there is hardly any mention of this county in the works of Genth or Kerr,* and comparatively few minerals are even mentioned by Dana as coming from this locality.

There is reason to think that the richness of the monazite in thoria is greater than in soft material, which, during the process of natural concentration, is more easily and quickly disintegrated than the harder and poorer rock.

I am much inclined to suppose also some correlation between the lens-formation and the curious occurrence in the same districts of corundum, beryl and plumbago, all of which are found in "pockets."

The production of monazite for the trade takes place in a very crude way, the absence of all sizing in connection with the concentration being the greatest fault. The operation consists in digging the gravel, shovelling it on an iron plate perforated with ½-inch or ⅜-inch holes, and stirring what passes this screen with a shovel, and much water, into a box about 15 inches wide and 6 to 8 feet long. The coarse gravel is thrown

* *Mineral Localities of North Carolina*, 1885.

away; the sand is worked through the box with shovels, somewhat in the manner of the German *Kehrheerd*, in which operation the lighter minerals are run off as tailings.

The absence of sizing causes all the finest sand to run off also, so that, of the product saved, generally nothing will pass a sieve of 40 to 50 mesh. At the same time the coarsest sand, of nearly ¼-inch diameter, runs off also, with the garnets.

On an average, the product contains about 80 per cent. of monazite. Exceptionally, material carrying 90 or even 95 per cent. is shipped; but there is also some that contains no more than 50 per cent. Sands of between 90 and 100 per cent. command 10 cents per pound; from 80 to 90 per cent. the price is 8 to 9 cents; from 70 to 80 per cent., 6½ to 7 cents; from 60 to 70 per cent., 5½ to 6½ cents, and below 60 per cent., 4 to 5½ cents.

The trade was pretty regular until April or May, 1895, when some sharp speculators caused a great disturbance, which greatly demoralized the business, especially in Europe, and allowed Brazilian sands to come in. At the same time the production of thorite in Norway increased very much, so as to make it difficult now to predict what will become of the whole industry in North Carolina. At present, only the Welsbach Incandescent Light Company and a German firm are buying monazite in much reduced quantities.

In connection with the curious and interesting formation of Augengneiss, and its general bearing upon the relations between gneiss and granite, and upon the paragenesis of minerals contained in enclosures of granite in gneiss, the following illustration may have collateral significance.

In May, 1895, I made a trip to Burnet, Texas, to examine the celebrated "Barringer Hill," in Llano county, about 15 miles west of Burnet. I traversed the whole country between the hill and Burnet on different roads, and found everywhere an extremely coarse-grained granite; but I crossed some strata of a gneiss rich in mica, and perhaps containing hornblende. I may here just suggest that its formation might also belong to the "eye-gneiss;" in this case the "eyes" would have diameters of miles.

The granite contains many quartz-veins, from one inch to several feet in thickness, and the "Hill" shows a swell about 100

feet in diameter, in which the quartz-kernels, as well as those of orthoclase, reach the size of 6, 8 and more feet in thickness. Crystals of orthoclase of 1 foot and more, and crystals of quartz of 4 to 5 inches, are not rare. Very curious is the fact that the feldspars leave their impressions in the quartz, while the latter leaves its impressions in the feldspar, which shows that the crystallization was, on the large scale, entirely contemporary.

Here, also, the fresh red orthoclase contains nothing, but the kaolinized orthoclase contains, I may say, minerals of all the rare metals enumerated above as occurring in monazite, besides molybdenum, beryllium, zirconium, fluorine and argon. The mica occurs in flat masses, some inches in thickness and one or more feet long, mostly decomposed. The whole called to my memory vividly the discussion by Prof. Sandberger of the veins of the Kinzigthal, in Baden, in which he demonstrated that the decomposition of the micas was the source of the formation of the ore-deposits. In that case the fresh micas, and probably the feldspars also, contained all the constituents of the veins (lead, copper, nickel, cobalt, etc.), and the decomposed micas contained considerably less, or nothing at all, of these metals. I do not know whether any analyses have been made with adequate accuracy of the Burnet micas and feldspars. All the oxides of the rare metals may easily escape detection, especially if they occur in minute quantities. Some analyses of yttrialite, thorogummite, nivenite and fergusonite, published in the *American Journal of Science* in December, 1889, are given by Dana. Other analyses have not come to my notice.

The Theory and Practice of Ore-Sampling.

BY D. W. BRUNTON, ASPEN, COLORADO.

(Atlanta Meeting, October, 1895.)

THE object of the investigations and experiments here recorded was to obtain such information and data as would make it possible to determine the fineness to which crushing must be carried, in sampling gold- and silver-ores, in order to obtain results within an allowable limit of error.

TABLE I.

Mesh.	No. of Wire.	Size of Wire. Inches.	Space. Inches.	Contents of Cube. Cubic Inches.	Weight of Cube in Pounds. Pyrite. Sp. Gr. 5. Weight per cubic inch, .1803.	Argentite. Sp. Gr. 7.3. Weight per cubic inch, .2633.	Galenite. Sp Gr. 7.5. Weight per cubic inch, .2705.	Silver. Sp. Gr. 10.6. Weight per cubic inch, .3823.	Gold. Sp. Gr. 17.6. Weight per cubic inch, .6348.	Weight of a Sample equally as safe as an Assay Weight of $\frac{1}{10}$ A. T.
	Brass. London Gauge.									
120	42	.0040	.0043	.00000007950	.00000001434	.00000002094	.00000002150	.00000003040	.00000005047	.0049
100	40	.0045	.0055	.0000001664	.00000003000	.00000004381	.00000004500	.00000006360	.0000001068	.0064
80	38	.00575	.0068	.0000003076	.00000005545	.00000008098	.00000008319	.0000001175	.0000001949	.0118
60	35	.009	.0077	.0000004512	.00000008135	.0000001188	.0000001221	.0000001725	.0000002860	.0174
	Iron. Worcest'r Gauge.									
50	34	.010	.0100	.000001000	.0000001803	.0000002633	.0000002705	.0000003823	.0000006348	.0385
40	31	.013	.0120	.000001728	.0000003116	.0000004549	.0000004675	.0000006605	.000001095	.0712
30	28	.016	.0173	.000005204	.0000009384	.000001370	.000001408	.000001990	.000003300	.2002
24	26	.018	.0237	.00001326	.000002391	.000003492	.000003587	.000005069	.000008108	.5102
22	25	.020	.0255	.00001648	.000002972	.000004340	.000004459	.000006302	.00001045	.6339
20	24	.023	.0270	.00001968	.000003548	.000005182	.000005324	.000007524	.00001247	.7571
18	23	.025	.0306	.00002854	.000005146	.000007515	.000007720	.00001091	.00001809	1.098
16	22	.028	.0345	.00004106	.000007404	.00001081	.00001110	.00001570	.00002603	1.580
14	20	.035	.0364	.00004835	.000008717	.00001273	.00001308	.00001848	.00003065	1.860
12	19	.041	.0423	.00007585	.00001368	.00001997	.00002051	.00002899	.00004809	2.918
10	18	.047	.0530	.0001489	.00002684	.00003920	.00004027	.00005695	.00009440	5.728
9	17	.054	.0571	.0001863	.00003358	.00004904	.00005039	.00007121	.0001189	7.166
8	16	.063	.0620	.0002383	.00004297	.00006275	.00006447	.00009111	.0001511	9.169
7	15	.072	.0709	.0003558	.00006415	.00009368	.00009625	.0001360	.0002256	13.69
6	14	.080	.0867	.0006510	.0001174	.0001714	.0001761	.0002489	.0004128	25.05
5	13	.092	.1080	.001260	.0002271	.0003317	.0003407	.0004816	.0007986	48.46
4	12	.105	.1450	.003019	.0005497	.0008027	.0008246	.001165	.001933	117.3
3	10	.135	.1983	.007801	.001407	.002054	.002110	.002982	.001946	300.1
2	8	.162	.3380	.03861	.006963	.01017	.01014	.01176	.02448	1486.
1	3	.244	.7560	.04321	.07791	.1138	.1169	.1651	.2739	16620.
1. inch space.........	3	....	1.0000	1.0000	.1803	.2633	.2705	.3823	.6318	38470.
1.5 " "	1	....	1.5000	3.3750	.6085	.8886	.9129	1.2900	2.1420	129800.
2. " "	0	....	2.0000	8.0000	1.4424	2.1060	2.1639	3.0580	5.0780	307800.
2.5 " "	....	....	2.5000	15.6250	2.8170	4.1140	4.2265	5.9730	9.9180	601100.
3. " "	....	....	3.0000	27.0000	4.8680	7.1090	7.3030	10.3221	17.1390	1039000.
4. " "	....	....	4.0000	64.0000	11.5308	16.8500	17.3101	24.4607	40.6206	2462000.

In Table I. are given, for several of the commonly occurring minerals, the sizes and weights of the largest cubes which will pass through sieves of the various sizes in ordinary use, from 120-mesh down to 4-inch space. In determining the size of the space, and hence the dimensions of the cube, for sieves above 50-mesh, the wire has been taken as having the diameter given by the London or Old English gauge; this being the standard adopted by the Wire Cloth Manufacturers' Association for brass-wire cloth, which is generally used in sieves of these sizes. Below 60-mesh, iron-wire cloth is commonly used, and for these sizes the Worcester gauge, which is the manufacturers' standard for iron-wire cloth, has been taken. In all cases where cloth of a given mesh is manufactured from several sizes of wire, the calculations have been based on the largest of these sizes, since these are the ones commonly used in sieves for screening ores. The specific gravities assumed for the minerals are averages taken from Dana.

In the table, the numbers in the first column indicate the mesh; the second column gives the number of wire assumed; the third gives the diameter of the wire in inches, taken from the manufacturers' tables for the appropriate gauge; the fourth gives the size of the space, or, in other words, the dimensions of the hole in the screen in inches; and the fifth column gives the volume, in cubic inches, of a cube whose edge is equal to the side of this hole. The next five columns give the weight of such cubes for pyrite, argentite, galenite, silver and gold. The last column, headed "Weight of a sample equally as safe as an assay weight of $\frac{1}{10}$ A. T.," gives the weight of a sample to which the weight of 1 cube of the size in question bears the same proportion that the weight of 1 cube of 100-mesh material bears to the weight commonly taken for assay, *i.e.*, $\frac{1}{10}$ assay-ton or .0064 pound; or, in other words, the weights in the last column contain the same number of cubes, assuming them all to be of maximum size, that the assay-weight contains, if its particles are all of maximum size. If, then, in cutting down a lot of ore of a given size to the weight indicated in the last column of the table, we get in the sample 1 cube too many or too little, it will have the same influence on the sample as would 1 cube excess or deficit in the assay weight; so that if, with the given ore, it is necessary to crush the assay sample to 100-mesh

in order to obtain sufficiently accurate results, it is also necessary in sampling down a large lot of the same ore, containing particles of richest mineral of any given size, not to go, without re-crushing, below the weight given in the last column of the table for that size.

Table II. gives the calculated effect which 1 cube of various gold- and silver-bearing minerals, of the maximum size which can pass a 100-mesh sieve, would produce on the assay-value of

TABLE II.

RICHEST MINERAL.	Specific Gravity.	Contents of 100-mesh Cube. Cubic Milligramme.	Weight of 100-mesh Cube.		Per cent of Gold or Silver in Richest Mineral.	Assay Value of Richest Mineral.	Difference in Assay Value of Ore caused by 1 Cube more or less of Richest Mineral.
			Milli-gramm's.	Assay Tons.			
Silver Minerals.						Ounces.	Ounces.
Pyrite	5.	.002726	.01363	.000000467			
						50	.000234
						100	.000467
						250	.001167
						500	.002335
Galenite	7.5	.002726	.02045	.000000701			
						50	.000351
						100	.000701
						250	.001753
						500	.003505
Tetrahedrite	4.8	.002726	.01309	.000000449			
						100	.000449
						250	.001123
						500	.002245
						1000	.004490
Pyrargyrite	5.8	.002726	.01581	.000000542	59.8	17441	.09453
Hessite	8.5	.002726	.02317	.000000795	62.8	18316	.14561
Cerargyrite	5.4	.002726	.01472	.000000505	75.3	21962	.11091
Argentite	7.3	.002726	.01990	.000000682	87.1	25404	.17326
Native Silver	10.6	.002726	.02890	.000000991	100.0	29166	.28904
Gold Minerals.						Dollars.	Dollars.
Pyrite	5.	.002726	.01363	.000000467			
						50	.000234
						100	.000467
						250	.001167
						500	.002335
Chalcopyrite	4.2	.002726	.01145	.000000393			
						50	.000196
						100	.000393
						250	.000983
						500	.001965
Petzite	9.	.002726	.02454	.000000841	25.	145830	1.2264
Native Gold	17.6	.002726	.04798	.00000165	100.0	583320	9.6248

an ore, if in excess or deficit in an assay-sample of $\frac{1}{10}$ assay-ton. The weight and contents of the 100-mesh cube are based on the diameter given by the London gauge for No. 40 wire, as explained under Table I. The specific gravities, as in Table I., are averages taken from Dana's *Mineralogy*. The weights of the cubes are given in milligrammes and in deci-

mals of an assay-ton, the latter being obtained from the former by dividing by 29166.

If we have the weight of a cube in decimals of an assay-ton, by multiplying this weight by the value of the mineral in ounces per ton, we obtain a result indicating the contents of the cube in ounces when referred to an assay-sample of 1 assay-ton; and by multiplying this result by ten, we get the number of ounces per ton represented by 1 cube, with reference to an assay-sample of $\frac{1}{10}$ assay-ton. In the same way, in the case of gold-ores, the number of dollars per ton represented by 1 cube is obtained. The last column of Table II. gives these results, and it is evident that, with the silver-bearing minerals, in few cases would a single particle no larger than a 100-mesh cube, represent sufficient value to cause, when in excess or deficit in an assay-weight of $\frac{1}{10}$ assay-ton, an appreciable difference. Only in the case of low-grade ores would one particle of pyrargyrite, hessite, cerargyrite, argentite or native silver cause an error of 1 per cent. or more; but with some ores carrying petzite or native gold, a material error would be caused by 1 cube excess or deficit in the assay-sample.

In Table III. are tabulated the actual weights of the largest particles found in lots of pyrite, argentite-filings, galenite, silver-filings, native silver, and native gold, of 60-, 80-, and 100-mesh in size.

The particles weighed were obtained by screening the crushed or filed mineral first through a 60-mesh sieve, rejecting what remained on the sieve, and then screening what went through on an 80-mesh sieve—what remained on the 80-mesh sieve forming the 60-mesh material of the Table. The 80- and 100-mesh materials were obtained in a similar manner. After the lots were all prepared in this way, the largest particles in each were picked out with pincers, by the aid of a magnifying glass, and weighed; and the weights given in the table are those of the largest single particle. For comparison, the calculated weights of cubes of each mesh and mineral are given in the table, and also the ratio which the actual weight bears to the calculated weight. In no case is this ratio less than 1, showing that all the lots of mineral worked upon contained elongated particles, or else that the screens all contained larger spaces than the nominal mesh would indicate.

The pyrite and galenite were crushed to the required size in a mortar, while the argentite and one lot of silver were obtained in small particles by filing from pieces of the pure minerals with a coarse file. The native silver was obtained by grinding down a piece of rock containing it on a bucking-plate, and then washing away as much as possible of the gangue. The native gold was obtained by washing from sand containing it in fine particles.

The results exhibited in Table IV. may be compared with those of Table II., being obtained in the same way, except that the actual weights of the largest particles taken from Table III. are used, instead of the calculated weights considering the particles as cubes. Figures are also given for 60-, 80-, and 100-mesh, while Table II. gives them for 100-mesh only.

The last column of Table IV. shows that among silver minerals a single particle of argentite of 100-mesh size in excess or deficit in a $\frac{1}{10}$ assay-ton weight, would give an error of 1 per cent. in ore running 26 ounces per ton; while a single particle of silver would give the same error in ore running 190 ounces per ton. Among the gold-minerals tabulated only native gold would cause a material error; one particle excess or deficit in $\frac{1}{10}$ assay-ton causing an error of 1 per cent. in an ore running about $5200 per ton.

The experiments and calculations described, and a general consideration of the subject, indicate that the size to which ore must be crushed, for sampling, in order to come within an allowable limit of error will depend upon:

1. The weight or bulk which the sample is to have. Evidently, the smaller the sample the finer the material must be crushed.

2. The relative proportion between the value of the richest mineral and the average value of the ore. If the average grade of the ore is high, in comparison with the grade of the richest mineral, a particle of richest mineral of a given size and value will have less percentage-effect on the sample than the same particle would have on the same amount of lower-grade ore; therefore, other conditions being the same, with high-grade ores we may crush more coarsely than with low-grade ones, and still keep within the same percentage of error; while if the richest mineral is of comparatively high grade, a particle

TABLE III.

	Pyrite.			Argentite Filings.			Galena.			Silver Filings.			Native Silver.			Native Gold.		
	1.	2.		3.	4.		5.	6.		7.	8.		9.	10.		11.	12.	
	Actual Weight of Largest Particle in Milligrammes.	Calculated Weight of Cubes in Milligrammes.	Ratio, 1:2.	Actual Weight of Largest Particle in Milligrammes.	Calculated Weight of Cubes in Milligrammes.	Ratio, 3:4.	Actual Weight of Largest Particle in Milligrammes.	Calculated Weight of Cubes in Milligrammes.	Ratio, 5:6.	Actual Weight of Largest Particle in Milligrammes.	Calculated Weight of Cubes in Milligrammes.	Ratio, 7:8.	Actual Weight of Largest Particle in Milligrammes.	Calculated Weight of Cubes in Milligrammes.	Ratio, 9:10.	Actual Weight of Largest Particle in Milligrammes.	Calculated Weight of Cubes in Milligrammes.	Ratio, 11:12.
60-mesh	.07	.0367	1.9	.06	.0535	1.1	.21	.0550	3.1	.08	.0777	1.0	.....	.0777	.....	.78	.1291	6.0
80- "	.06	.0252	2.4	.05	.0368	1.4	.16	.0378	4.2	.07	.0535	1.3	.25	.0535	4.7	.36	.0888	4.1
100- "	.04	.0136	2.9	.03	.0199	1.5	.09	.0205	4.4	.06	.0289	2.1	.19	.0289	6.6	.26	.0480	5.4

TABLE IV.

Richest Mineral.	Actual Weight of Largest Particles.						Per cent. of Gold or Silver in Richest Mineral.	Assay Value of Richest Mineral.	Difference in Assay Value of Ore caused by 1 Particle more or less of Richest Mineral.		
	60-Mesh.		80-Mesh.		100-Mesh.						
	Mgrs.	Assay Tons.	Mgrs.	Assay Tons.	Mgrs.	Assay Tons.			60-Mesh.	80-Mesh.	100-Mesh.
Silver Minerals.								Ounces.	Ounces.	Ounces.	Ounces.
Pyrite	.07	.00000240	.06	.00000206	.04	.00000137	......				
								50	.00120	.00103	.00068
								100	.00240	.00206	.00136
								250	.00600	.00515	.00342
								500	.01200	.01030	.00685
Galenite	.21	.00000721	.16	.00000548	.09	.00000309	......				
								50	.00360	.00274	.00155
								100	.00720	.00548	.00310
								250	.01800	.01370	.00775
								500	.03600	.02740	.01550
Argentite Filings	.06	.00000206	.05	.00000171	.03	.00000103	......	25404	.52332	.43410	.26166
Silver Filings	.08	.00000274	.07	.00000240	.06	.00000206	......	29166	.79914	.69998	.60082
Native Silver	....	....	.25	.00000855	.19	.00000651	......	29166	....	2.4934	1.8987
Gold Minerals.								Dollars.	Dollars.	Dollars.	Dollars.
Pyrite	.07	.00000240	.06	.00000206	.04	.00000137	......				
								50	.0012	.0010	.0007
								100	.0024	.0021	.0014
								250	.0060	.0052	.0034
								500	.0120	.0103	.0068
Native Gold	.78	.00002674	.36	.00001234	.26	.00000891	......	583320	155.9797	71.9817	51.9738

of it, of given size, will have a greater effect on the sample than if it is of low grade, and this will necessitate finer crushing.

3. The specific gravity of the richest mineral. The higher the specific gravity of the richest mineral the greater the value contained in a particle of given size and grade, and hence the greater the influence of such particle on the sample; from which follows the necessity of keeping down the size of the largest particles by finer crushing than is required when the richest mineral is of lower specific gravity.

4. The number of particles of richest mineral which are likely to be in excess or deficit in the sample is evidently an important factor; a liability to a large number necessitating especially fine crushing. But such liability can result only from imperfect mixing, and for material mixed with average thoroughness this number must be small.

The relative proportion, by weight or bulk, of the richest mineral and the low-grade or average ore, will not of itself affect the size to which the ore must be crushed. For we assume that so far as forming an excess or deficit in the sample is concerned, all particles composing the lot of ore stand upon an equality, uninfluenced by size or other characteristics; and also that the number of particles in excess or deficit is limited. Therefore, assuming a lot of ore properly crushed and mixed, the principal effect which a large proportion of richest mineral has, is to increase the proportion of maximum-sized particles of richest mineral with reference to the whole number of particles composing the lot, or in other words, to increase the probability of the occurrence of maximum variations. But the limit of the magnitude of these variations is just the same as when only a small proportion of the richest material is present.

Inasmuch as the effect of a particle in excess or deficit in a sample depends only on its weight and value, as compared with those of the sample itself, it is evident that the size or weight of the lot of ore from which a sample is taken, or in other words the proportion of the lot of ore taken as a sample, does not directly enter into the question of determining the maximum size which the ore may have.

If we have a sample of ore of any weight, containing particles of any size, and add to it a particle of any size and value, it is clear that after such addition the total value in the sample

will deviate from that given by its former average grade, by an amount equal to the value contained in the particle added, less the value contained in a particle of the same weight, but of the grade first possessed by the original sample. Also, if we remove any particle from such sample its total value will then deviate from that given by its former average grade, by the same amount.

Let W = weight of sample in pounds.
k = grade of richest mineral in ounces per ton.
c = average grade of ore in ounces per ton.
s = specific gravity of richest mineral.
n = number of maximum-sized particles of richest mineral in excess or deficit in sample.
f = a factor expressing the ratio of the actual weight of richest mineral in the largest particles which will pass a screen of given size, to the weight of the largest cubes of richest mineral which will pass the same screen.
p = allowable percentage-error in sample.
D = diameter, in inches, of the holes in screen, or the nominal diameter to which the ore is crushed.

Then, the weight of a cubic inch of water being .036 pounds, the weight of a maximum *cube* of richest mineral is $.036\ s\ D^3$; and the actual weight of a maximum *particle* of richest mineral is $.036\ f\ s\ D^3$.

The silver contents of a maximum *particle* of richest mineral are $\dfrac{.036\ f\ s\ k\ D^3}{29166}$.

The silver contents of a particle of same weight but of average grade are $\dfrac{.036\ f\ s\ c\ D^3}{29166}$.

Then the deviation caused by one maximum particle of richest mineral in excess or deficit in sample is

$$\frac{.036\ f\ s\ k\ D^3}{29166} - \frac{.036\ f\ s\ c\ D^3}{29166} = \frac{.036\ f\ s\ D^3}{29166}(k - c).$$

The silver contents of the sample should be $\dfrac{W c}{29166}$, and the allowable error is $\dfrac{W c p}{29166 \times 100}$. If this error is caused by n

maximum particles of the richest mineral in excess or deficit, we have:

$$\frac{.036\, f\, s\, n\, D^3}{29166}(k - c) = \frac{W\, c\, p}{29166 \times 100},$$

from which $3.6\, f\, s\, n\, D^3\, (k - c) = W\, c\, p.$

$$D^3 = \frac{W\, c\, p}{3.6\, f\, s\, n\, (k - c)}, \qquad (1.)$$

$$D = .65\, \sqrt[3]{\frac{W\, c\, p}{f\, s\, n\, (k - c)}}, \qquad (2.)$$

In this formula, p can be made $= 1$, as 1 per cent. is about as close as the assay work can be done. Table III. gives values of f for a number of common minerals, for small sizes, while for large sizes, f is probably rarely greater than 1. But in many cases where the ore is in large pieces the value of f would be less than 1, because only a portion of a piece would be composed of the richest mineral itself, in a pure condition. The value of n depends on the thoroughness of mixing, and data for computing it could be obtained by experimenting with well-mixed material.

Data for obtaining the value of n for a special case are given in Table V. which shows the results of assays made on four samples of rather high-grade ore carrying native silver. Inspection of the table shows that the results of sample No. 4 gave the greatest average deviation from the mean, and it is therefore taken as a basis for computing n. Assuming that the mean result gives the most nearly correct value for sample No. 4, we find the average error or deviation to be $\frac{88}{100}$ of 1 per cent. The material was crushed for assay to pass a 100-mesh screen, so the value of D in the formula is .0055 inches (see Table I.). One-tenth assay-ton was taken for each assay, making $W = .0064$ pounds.

We then have:

$W = .0064.$
$k = 29{,}166.$
$c = 568.$
$s = 10.6.$
$f = 6.6$ (from Table III.).
$p = .88.$
$D = .0055.$

Placing the formula in the form,

$$n = \frac{.278\ W c p}{f s D^3 (k - c)}, \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (3.)$$

and substituting these values:

$$n = \frac{.278 \times .0064 \times 568 \times .88}{6.6 \times 10.6 \times .0055^3 \times (29{,}166 - 568)} = 2.64.$$

TABLE V.

SAMPLE.	No. 1.		No. 2.		No. 3.		No. 4.	
	Results of Assays.	Deviations from the Mean.	Results of Assays.	Deviations from the Mean.	Results of Assays.	Deviations from the Mean.	Results of Assays.	Deviations from the Mean.
1	577.7	2.7	560.0	3.9	583.0	3.7	563.7	4.4
2	568.9	6.1	563.3	.6	585.5	1.2	575.6	7.5
3	578.5	3.5	568.4	4.5	591.5	4.8	565.0	3.1
	1725.1	12.3	1691.7	9.0	1760.0	9.7	1704.3	15.0
Means.........	575.0	....	563.9	....	586.7	....	568.0	.....
Average Deviations....		4.1	....	3.0	....	3.2	....	5.0

Table VI. exhibits another series of results from which the value of n may be deduced. The experiments by which these results were obtained were conducted as follows:

A quantity of argentiferous limestone carrying about 15,000 ounces per ton, in native silver, was crushed to pass an 80-mesh screen, and then thoroughly mixed with such a quantity of an ore carrying about 15 ounces per ton, as to bring the grade of the mixture to about 300 ounces. This mixture was divided into two parts; the first was assayed without further crushing, and the results are given in column 2 of the table. The second portion was ground in an agate mortar until it would pass a 120-mesh screen; the metallics being saved, and great care taken to prevent loss. The coarse pulp was weighed before grinding in the mortar, and the fine pulp and metallics afterwards, and the loss was .43 of 1 per cent. This fine pulp was then assayed, and the results are given in column 5 of the table. To obtain the average value of the ore, the value of the

metallics remaining on the screen must be added to the mean of the results in this column. If the result thus obtained (307.28 ounces) is assumed as the most probable value of the ore, and the deviations from it of the results in column 2 are obtained, their average, expressed in percentage of the value of the ore, furnishes a value of p in the formula, and, together with the other data of the table, gives means of computing n.

The quantities entering into the computation, then, are:

$W = .0064$, as 1/10 A. T. weights were used for assay.
$k = 29{,}166$.
$c = 307.28$.
$s = 10.6$.
$f = 4$.
$p = 1.83$, from column 4, Table VI.
$D = .0064$, measured.

Substituting in the formula $n = \dfrac{.278\, W c\, p}{f s D^3 (k - c)}$,

$$n = \frac{.278 \times .0064 \times 307.28 \times 1.83}{4 \times 10.6 \times .0064^3 \times (29{,}166 - 307.28)} = 3.14.$$

If it is assumed that 3 is a safe value for n, and we make $p = 1$, the formula, [1], deduced on page 836 becomes

$$D^3 = \frac{W c}{3.6 \times 3 f s (k - c)} = \frac{W c}{10.8 f s (k - c)}, \quad . \quad (4.)$$

and

$$D = .45 \sqrt[3]{\frac{W c}{f s (k - c)}} = .45 \sqrt[3]{\frac{W}{f s \left(\frac{k}{c} - 1\right)}}, \quad . \quad (5.)$$

or

$$D = .45 \times \frac{1}{\sqrt[3]{f s}} \times \sqrt[3]{W} \times \frac{1}{\sqrt[3]{\frac{k}{c} - 1}}, \quad . \quad (6.)$$

If now we consider a group of minerals of specific gravity, for example, 5.5, or slightly above or below, and for which f may be given a general value of, say 4, for small sizes, the term $\dfrac{1}{\sqrt[3]{f s}}$ in the above formula, [6], becomes constant, and we have $D = R \times \sqrt[3]{W} \times \dfrac{1}{\sqrt[3]{\frac{k}{c} - 1}}$, where R is a constant, depending

TABLE VI.

1.	2.	3.	4.	5.	6.
SAMPLE.	Assay Results. 80-Mesh Material.	Deviations from the Mean.	Deviations of Results in Column 2 from Mean Value of Mixture as Determined from 120-Mesh Material.	Assay Results. 120-Mesh Material.	Deviations from the Mean.
No. 1	299.8	7.68	7.48	301.9	.32
2	303.5	3.98	3.78	302.7	1.12
3	319.5	12.02	12.22	302.3	.72
4	302.9	4.58	4.38	297.0	4.58
5	312.5	5.02	5.22	305.3	3.72
6	306.7	.78	.58	300.3	1.28
	1844.9	34.06	33.66	1809.5	11.74
				301.58	....
Metallics	....	....	....	5.7	....
Means	307.48	....	....	307.28	....
Average Deviations	....	5.68	5.61	....	1.97
Average Percentage Deviations	....	1.85	1.83	....	.65

on the values of f and s. If, now, a value is assumed for W, the formula becomes $D = R_1 \times \dfrac{1}{\sqrt[3]{\dfrac{k}{c}} - 1}$, and the locus of this equation will be a curve whose co-ordinates may be made to indicate values of D and of $\dfrac{k}{c}$.

The curves shown in Plates I. and II., have been constructed in this way, and the following table shows the values taken for the constants in the formula for each of the 4 groups which have been assumed.

The weights of sample chosen form a geometrical progression whose ratio is 10. The diameters for samples of intermediate weights may be obtained by interpolating roughly, by eye, between the proper curves; or the diameter may be taken

	Sp. Gr. s.	f.		Weights.
		Small Sizes.	Large Sizes.	
Group I........................	5.5	4	1.5	Curve 1, .006 2, .06 3, .6 4, 6. 5, 60. 6, 600. 7, 6000. 8, 60000.
Group II.......................	8.0	4	1.5	Same as Group I.
Group III.......	10.6	6	1.5	Same as Group I.
Group IV......................	18.	6	1.5	Same as Group I.

from the curve next lower than the actual weight of sample. This will have the effect of lowering the limit of percentage-error. Thus, if for a 6000-pound sample the diameter is taken from the 600-pound curve, the limit of error will be $\frac{1}{10}$ of 1 per cent., instead of 1 per cent. The values of f selected are, for the first two groups, 4 for sizes under .025 inch diameter, or about 22-mesh, and 1.5 for larger sizes. Two curves would thus be obtained, which in the diagrams are graduated into each other, so that for sizes above .025 inch diameter the value of f decreases rapidly to 1.5. For the second two groups the values of f assumed are 6 and 1.5. The latter figure is too large for the largest pieces which occur in practice, while a smaller value, used for all sizes below .025 inch diameter, would give results too high for sizes from .025 inch to perhaps .5 inch or 1 inch. Also, a continual change of the value of f increases greatly the labor of calculating the curves. So $1\frac{1}{2}$ has been taken as being on the side of safety for all sizes for which it has been used. The error thus introduced in even the largest sizes is not very great, as the diameter varies with the reciprocal of the cube root of f, and not inversely with the factor itself.

If sufficient experimental data were available, it would probably be possible to deduce an approximate expression for f, in

terms of D and certain numerical constants, which would be safe for all minerals. This value of f could then be substituted in equation (4), and the formula for the diameter deduced from this would be of general application for all minerals having the same, or nearly the same, specific gravity; while curves calculated from such a formula would be closer approximations for all sizes than those calculated with a constant value for f.

The range for specific gravities is from 5.5 to 18, which would include all the valuable minerals commonly occurring in ores. The effect of specific gravity is indicated by the distance between successive curves for the same weight of sample, provided the diameter .025 is not crossed, in which case the change in the value of f must also be considered.

In Plates I. and II., ordinates or vertical distances indicate diameters, and abscissæ or horizontal distances indicate values of $\frac{k}{c}$ in ounces or dollars per ton, or in percentages. That portion of the curves which includes diameters greater than $\frac{1}{10}$-inch is shown on Plate I., while Plate II. shows the portion including diameters less than $\frac{1}{10}$-inch. In order to increase the accuracy with which the diameters may be taken from Plate II., the scale for ordinates is much larger than in Plate I. The scales for both ordinates and abscissæ are, however, clearly indicated on both plates. There are eight curves, (one for each assumed weight of sample), for each of the four groups based on specific gravity. All curves of the same specific gravity are drawn in the same character of line, that is, either dots, dashes, dashes and dots, or continuous line. The outside columns in the margins contain scales for the co-ordinates of the curves. The figures in the middle columns indicate the specific gravities used in computing the curves opposite which they are placed, and the inner columns in the margins contain, opposite each curve, figures indicating the weight for which the curve is calculated. To illustrate the use of the diagram, suppose we have an ore whose average value is about 50 ounces, but which contains pyrites running up to 2500 ounces per ton, and suppose a sample of 500 pounds is to be taken. The value of $\frac{k}{c}$ is $\frac{2500}{50} = 50$. Looking in the inside column of the right hand margin of Plate I., the nearest weight to 500 pounds is 600 pounds. The specific

gravity of pyrites is 5, and in the middle column we find the nearest value to this (5.5), opposite the continuous-line curve; following this curve until we come to the ordinate marked 50 on the top and bottom scales, we find the curve intersecting this ordinate at a distance on the vertical scale of .50. The next lowest continuous-line curve intersects this ordinate at about .25. Hence if we crush to .25 inch diameter the limit of error will be about $\frac{1}{10}$ of 1 per cent., while if we crush to .50 inch it will be slightly over 1 per cent. Suppose this same ore contained native silver, $\frac{k}{c}$ would then be about 600, and the specific gravity, 10.6. In the same way as before (but using the curve drawn in dashes), we find that for a 600-pound sample the diameter should be .175 inch. Making some allowance for the fact that few or no pieces of silver would be as large as .175 inch diameter, we could no doubt crush a 500-pound sample to this size or a little larger with perfect safety. For another example, take an ore averaging 30 ounces, carrying galenite running as high as 3000 ounces per ton; to find the diameter to which it should be crushed for assay. Here $\frac{k}{c}$ is 100. The specific gravity of galenite is 7.5, or, taking the nearest on the diagram, 8. An assay-ton is .0064 pounds, the next lower weight on the diagram being .006. We find the intersection of the dash-and-dot curve marked .006 with the ordinate numbered 100 opposite the diameter .0057, which corresponds to about 80-mesh. Suppose this ore contained native silver, $\frac{k}{c}$ would then be 1000, and the specific gravity would be 10.6, from which we find the diameter .0020 inches, which is finer than the opening of a 120-mesh sieve.

Table VII. exhibits the results obtained from a series of twelve assays, each on mixtures of native silver with a low-grade ore carrying 15.5 ounces silver per ton.

Mixture No. 1 was made by mixing this low-grade ore with filings from cupelled silver. These filings had been screened through a 100-mesh sieve, and then screened on a 120-mesh sieve, so they were composed only of particles between the two sizes. In the same way, Mixture No. 2 contained particles ranging from 120- to 140-mesh, while Mixture No. 3 contained particles ranging from 140-mesh down. The average percentage-

TABLE VII.

Mixture.	No. 1. 100-Mesh.		No. 2. 120-Mesh.		No. 3. 140-Mesh.	
	Results of Assays.	Deviati'ns from the Mean.	Results of Assays.	Deviati'ns from the Mean.	Results of Assays.	Deviati'ns from the Mean.
1	299.7	2.63	299.2	1.73	294.5	6.88
2	296.6	.47	306.7	9.23	307.8	6.42
3	309.8	12.73	300.6	3.13	305.1	3.72
4	293.0	4.07	294.3	3.17	305.8	4.42
5	299.3	2.23	296.8	.67	308.2	6.82
6	295.0	2.07	290.7	6.77	297.0	4.38
7	288.5	8.57	306.3	8.83	292.7	8.68
8	297.6	.53	297.3	.17	298.4	2.98
9	302.2	5.13	297.7	.23	307.0	5.62
10	288.4	8.67	289.8	7.67	302.6	1.22
11	293.0	4.07	299.0	1.53	301.1	.28
12	301.7	4.63	291.2	6.27	296.3	5.08
	3564.8	55.80	3569.6	49.40	3616.5	56.50
Means	297.07	....	297.47	....	301.38	....
Average Deviations	....	4.65	....	4.12	....	4.71
Average Percentage Deviations	....	1.57	....	1.38	....	1.56

deviations from the mean, in the three cases, indicate that the three sets of results are of substantially the same degree of reliability, and not that, as might be expected, the most reliable results were obtained from the 140-mesh material. Examination of the ordinary 100-, 120- and 140-mesh screens shows that, owing to irregularities in the mesh, there are openings in the 120- and 140-mesh screens nearly or quite as large as those in the 100-mesh, and from this we would be led to expect just such results as are presented in Table VII. These expectations are also confirmed by the following figures showing the average weights of the largest particles of native silver which had passed through screens of the sizes in question.

Mesh of screen, ...	100	120	140
	Mgrs.	Mgrs.	Mgrs.
Weight of largest particle, ...	.06	.071	.04

The principal error in sampling, as usually practiced, arises from the use of screen-cloth irregularly spaced and partially

worn, and from cutting samples down too far without recrushing. It was only after the most careful search among the products of the leading wire-manufacturers that brass screen-cloth up to 100-mesh, uniformly spaced and rolled as smooth as glass, could be found. This latter is an extremely important point, as it is common for assayers to put a small weight into the screen while sifting, in order to keep the meshes from becoming clogged, and unless the wires are rolled down absolutely smooth, the projecting wires are worn off by this weight, thereby increasing the size of some of the openings and rendering the screening irregular and unsafe. The results of the investigations recorded in this paper show how absolutely necessary it is that ore-samples should be recrushed after each successive "cutting-down," so that as the sample diminishes in weight, there may be a nearly constant ratio between the weight of the sample and that of the largest particle of ore contained therein.

Mining Titles on Spanish Grants in the United States.

BY R. W. RAYMOND, NEW YORK CITY.

(Atlanta Meeting, October, 1895.)

THE "law of the apex," with its vague and impracticable "extra-lateral rights," is defended—if it is defended at all—on the ground that it is required by the peculiar conditions of the mineral deposits or of the mining industry in those States and Territories in which it is operative. As I have repeatedly attempted to show, in papers read before the Institute, and most recently in an article on "The Relation of the Mining Law of the United States to the Development of its Mineral Resources," published in Mr. Rothwell's *Mineral Industry* for 1894, there is now (whatever may have been the case formerly) no peculiarity of conditions in the regions referred to which warrants the subjection of their mining industry to a burden so vexatious and injurious as this law imposes. I have called attention to the absolute freedom from expensive litigations over mining titles enjoyed by all other States in the Union and all other regions in the world where the absurd extra-lateral right does not exist.

Certainly, it will not be contended that the metalliferous deposits of our public domain in the West are radically different in kind or form from those which are successfully exploited elsewhere under the operation of the simple principles of common-law boundaries. Even in the territory still afflicted with "the law of the apex," that law is not applied to coal-mines, though in some cases coal-seams are mined which dip more steeply than many of the most productive "lodes." It might be supposed that the absence, in such cases, of all litigation concerning the boundaries of mining rights would be instructive and suggestive to the owners of metal-mines in the same regions, leading them to desire a similar certainty of ownership for themselves.

Yet it remains true, on the whole (though for a smaller portion of the citizens than formerly), that proposals for a change in the existing law are unpopular in the West. The opinion of intelligent lawyers, I think, has come very generally to favor the adoption of the vertical boundaries which work so well everywhere else in the world. But lawyers, as such, have no direct interest in putting an end to the system which gives them so much profitable business, and although I feel sure that the members of that profession, if consulted, would frankly express the view I have ascribed to them, it is scarcely to be expected that they should engage actively in the agitation of the subject. The late Mr. Symes, of Denver, entering Congress as a Representative from Colorado, after a long experience as counsel in mining litigation, introduced in the House of Representatives a bill providing for the future location and sale, on the public domain of the United States, of mining claims with vertical boundary-planes, and encountered, as a consequence, the vehement disapproval of his constituents, or, at least, of such of them as chose to express themselves through mass-meetings. The bill failed, as had failed also the earlier bill to the same effect, framed by the Public Lands Commission in 1880.* In both instances the cause of failure was the hostility to the proposed reform exhibited by the communities affected. It is not my purpose here to analyze in detail the sources and reasons of

* This commission consisted of Messrs. J. A. Williamson (then Commissioner of the General Land Office), Clarence King (then Director of the U. S. Geological Survey), A. T. Britton, Thomas Donaldson and J. W. Powell.

this hostility; but I may be permitted to offer a very general statement.

We may roughly divide the western communities in which this sentiment is manifested into three classes: Those who own mines; those who hope to own locations of speculative value; and those who profit indirectly by the mining industry. The first class would not be affected by a change in the law, since such a change would not affect vested rights. The second class, comprising a much larger part of the population, is afraid that the proposed change in the law would diminish the chances and rewards of the prospector; and the third class, impressed by this attitude on the part of the second class, and feeling that anything which hinders activity in prospecting will react unfavorably upon general trade and the value of real estate, gives at least a passive support to the demand of the prospectors.

We have, then, to deal with the feeling of the prospectors as the true kernel of the difficulty; but in this class must be included not merely the individuals who are, at any given time, ranging the mountains in search of promising outcrops, but also those who are, as partners, paying their expenses ("grub-staking" them), and also those who are earning wages, but who intend to go prospecting when they have saved enough to furnish them for such an excursion. To take the State of Colorado as an example, I think I may say that almost all the miners working for wages are also from time to time prospectors, and that almost all the merchants are, from time to time, supporting prospectors or acquiring property in undeveloped mining claims: so that the interest of the prospector is really shared by a large part of the population.

I must include in this class also (although they are not worthy to be ranked with honest explorers) those who make locations on the outskirts of valuable claims for the express purpose of piracy or blackmail, and those who support them by furnishing for vexatious litigation the means which would not be forthcoming for honest mining. These people thrive on the ambiguities and complications of the mining law. If it were made simple and clear, their occupation would be gone. There is no doubt, therefore, on which side they stand. But, disregarding them, let us consider the interest of the *bona fide* prospector.

Since it happens but seldom that a mine pays "from the

grass-roots down," the prospector must be aided, in most instances, with the capital necessary for development. Even when his own labor in prospecting-pits and drifts is repaid by the ore incidentally produced, the time soon comes when capital is needed for operations on a large scale. As a practically universal rule, prospectors work in the hope of selling their claims or interests therein. If the law should be so changed as to give them claims twice as large as they can now locate, but, at the same time, forbid transfer of title by sale, the business of prospecting would be ruined. Everybody knows that. Moreover, the sellers of mining claims to non-resident purchasers have found out that U. S. patents are required to make the titles acceptable. Everybody knows *that*, too, and everybody has submitted to it. But under the present law, the United States patent does not give a satisfactory title. It is "iron-clad" and invulnerable as to what it actually grants, but what that is it does not set forth, so that a purchaser can know what he is buying. The consequence is that, by reason of this cloud upon title, mining claims cannot be sold so easily or at as good prices as if the title were more completely defined. This is a distinct injury to the prospector more than to any other party concerned. The other victims have their escape, if not their remedy. The eastern or English company which has bought a good mine, only to lose its profits in litigation, can buy mines in some better-governed country; but the prospector who finds it harder and harder to secure foreign capital, can only give up his barren trade. Just now immense sums are forthcoming for the purchase of mines in South Africa and West Australia. Can there be any doubt that the investment in those regions of hundreds of millions, without a single instance of litigation over boundaries, such as blights our West, has been an object-lesson of much significance to English capitalists? Can there be any doubt that if the Witwatersrand had been located in Colorado, instead of South Africa, there would have been, long before this, a crop of lawsuits over apexes, side-lines, strikes, dips, faults, vein-matter, assays, theories of formation and subtle expert distinctions which would have choked the new industry and corrupted the atmosphere surrounding it?

Such considerations ought certainly to have weight with our prospectors; but they are short-sighted, and need to have object-lessons nearer than South Africa or West Australia, or even

the Carolinas and Georgia, or Michigan and Wisconsin, or Missouri, or British Columbia. I am very glad, therefore, that the practicability of the "square location" can be illustrated in the heart of Colorado.

Under the famous California decisions, confirmed by the U. S. Supreme Court, a patent from the United States, issued in confirmation of a land-grant from the former government of Mexico, invests the patentee with the ownership of the precious metals which the land may contain. The owners of these old Spanish grants have, therefore, the power to make such regulations for the sale or working of mines as they may deem advisable. In most cases, I believe, they have followed the apex-feature of the United States law, fearing to arouse local prejudices by departing from local usage. This is the case, if I am not mistaken, upon the Cerrillos and Maxwell grants, in New Mexico. But the proprietors of another similar estate have adopted what seems to me to be a wiser policy, as the following proclamation shows:

NOTICE TO MINERS AND PROSPECTORS.

The western slope from the summit of the range of mountains forming the eastern boundary of the San Luis Valley, and extending southerly from a point in the neighborhood of old Fort Garland, into New Mexico, is included in what is known as the Sangre de Christo grant, and is *patented* land.

Beginning at a point nearly opposite the town of San Luis, the county seat of Costilla county, Colorado, this range of mountains, to the summit thereof, and in some places beyond the summit, and to the boundaries of the Maxwell grant on the east, is the property of the United States Freehold Land and Emigration Company, the title of said company being perfect and undisputed. Said range of mountains, from specimens of float ore, found from time to time, is known to contain numerous mineral-bearing veins and lodes, and is believed to be rich in precious metals.

The United States Freehold Land and Emigration Company now throws open to prospectors its lands in said range, in Colorado and New Mexico, upon terms more reasonable than those offered by the United States Government. (See Mining Regulations below.)

The country is very accessible, and may be easily reached by good roads from Garland, on the Denver and Rio Grande railroad. For particulars, apply to

E. C. VAN DIEST,
Manager,
San Luis, Colorado.

Mining Regulations for Prospectors and Intending Locators Upon the Lands of the United States Freehold Land and Emigration Company.

Notice is hereby given that the lands of the United States Freehold Land and Emigration Company, situated in the County of Costilla and State of Colorado, and County of Taos, Territory of New Mexico, are thrown open to prospectors

for the discovery, location and actual ownership of mineral-bearing veins, lodes and deposits (except such as are chiefly valuable for coal and iron), under the following regulations:

SECTION I. The discoverer of any mineral-bearing lode, vein or deposit within the limits of the estate now owned by the company may locate a claim upon said vein, lode or deposit, 1500 feet in length along the supposed course of said vein, lode or deposit, by 150 feet on each side of the point of discovery of such vein, lode or deposit, by posting a notice at the point of discovery, giving the name of the claim, the number of feet claimed in each direction from the point of discovery, the name or names of the locator or locators, and the date of discovery, and by filing a copy of such notice in the office of the company at San Luis, Colorado. (No fee for filing such notice will be charged during the years 1891 and 1892, but on and after January 1, 1893, a fee not exceeding three dollars will be charged for each notice.) Blank forms of notices will be furnished to intending locators free of charge.

SEC. II. The extensions for 1500 feet in length by 300 feet in width along the vein, lode or deposit, each way from the end-lines of every claim so located, shall remain the property of the United States Freehold Land and Emigration Company, and shall not be subject to location, under these regulations, but may be negotiated for and purchased upon terms mutually satisfactory to the company and the purchaser.

SEC. III. The locator or locators, upon filing the notice of their claim provided for in Section I., shall, within sixty days from the date of such filing, determine as nearly as may be the extent and course of the vein, lode or deposit claimed by them, and mark the bounds of the claim upon the ground, by stakes or other readily distinguished monuments, one of such stakes or monuments to be set as nearly as may be at each corner and one at the center of each side-line of the claim, and shall, within the time aforesaid (60 days), file in the office of the company at San Luis, Colorado, a plat of the said mining-claim, showing the supposed course of the vein, lode or deposit, and the number of feet claimed along the strike of the same in each direction from the point of discovery, which plat must connect said mining-claim with the surveys of the company, or with some locating monument to be established by the engineer of the company. (No fee will be charged for this filing.)

SEC. IV. Except as modified by the consent of the company, or by prior contracts or conveyances, each claim shall be 1500 feet in length by 300 feet in width. The side-lines of the claim must be parallel with each other, and except as provided above in this section, must be equidistant from the point of discovery, but need not be equidistant from the center of the lode, vein or deposit. The end-lines must be parallel with each other.

SEC. V. Within one year from the date of filing the notice of location in the office of the company, as provided in Section I., the locator or locators shall do at least one hundred dollars' worth of work upon the lode or vein, and shall make written application to the company to purchase said mining claim and have an official survey of the same made by the company's surveyor, and shall deposit in the office of the company fifty dollars in payment for such survey.

SEC. VI. Upon completion of the survey provided for in Section V., the company will execute a deed to the surface ground of the mining claim, together with all the minerals contained therein (except coal and iron, as aforesaid), upon the payment to the company of ten (10) dollars per acre for the land contained in said claim, within thirty days from the completion of said survey.

No right to mine beyond the boundaries of said claim (either side-lines or end-lines) will be given.

SEC. VII. Priority of location shall govern all awards, but in case of adverse claimants to the same lode, vein or deposit, the company will not undertake to adjudicate the rights of the respective claimants, but will recognize such rights as may be successfully established in the courts, in proceedings between the claimants, provided these regulations have been complied with by the successful litigant.

SEC. VIII. Failure on the part of any locator or locators to comply with any of the foregoing regulations shall work a forfeiture of all rights, and shall be deemed an abandonment of the mining claim.

SEC. IX. These regulations shall not be construed so as to prevent the company from disposing of its unlocated lands as it may deem fit.

UNITED STATES FREEHOLD LAND AND EMIGRATION COMPANY.

Mr. E. C. Van Diest, the manager of this estate, who is a member of the Institute, writes me that two years' experience under these regulations has shown them to work admirably. More than three hundred claims have been located under them, and Mr. Van Diest says that he has met but two persons who objected to the rules. The company provides forms of location-certificates and records them in its office. Moreover, the company, as I am informed, maintains a map, showing at a glance what mining-claims have been sold or are held under location. When any claim has been forfeited (under Section VIII.) prior to its final sale, the map is altered accordingly. I understand that blue prints of this map are made monthly and furnished to applicants, so that any person desiring to make a mining location can ascertain at once what land is free for the purpose. This simple system is in striking contrast with the blundering confusion of the United States law. According to that law, the making of a mining location, in compliance with certain rules (which include no notice whatever to the officers of the United States, but only a record with a local State, territorial or county officer, or a "mining recorder"), immediately withdraws from the public domain the land concerned. Not until proceedings for United States survey and patent are commenced (and that need never be done if the locator prefers to hold by annual "assessment-work") does the United States Land Office receive any notice of the location; and the maps of the government to-day represent hundreds of thousands, perhaps millions, of acres as belonging to the public domain which have, in fact, been withdrawn from it and are held by parties unknown.

Uncle Sam doesn't know what he has to sell. All he can do for a proposing purchaser is to smile feebly and say: "So far as I know, the land belongs to me. But you must advertise for ninety days, and if nobody turns up to claim the land I will assume that it is mine and sell it to you."

In other words, the United States surrenders to locators on its domain rights and titles of which it makes no record. Such record as there is, is made and kept by officials not in its service.

Some of the provisions in the regulations quoted above seem to me unnecessary. I do not see, for instance, why the side-lines of a claim must be equidistant from the point of discovery, especially as that point is not required to be on the "apex," and the side-lines need not be equidistant from the center of the deposit or parallel with its course. The point of discovery is purely an accident. Why not sell locations of any desired size or shape up to a certain maximum area? It must be observed, however, that the company does, in fact, reserve the right to consent to modifications of form.

The final conveyance is made by warranty-deed; and I do not hesitate to say that such a deed, under the circumstances, conveys a better title than can be got by United States mineral-land patent in other parts of Colorado. It need scarcely be added that no lawsuits have yet arisen, on the Sangre de Christo grant, over the boundaries and nature of mining rights. Two disputes, which would doubtless have developed into promising suits under the United States mining law, were settled by simple reference to the records in the office of the company.

It is to be hoped that the experiment thus inaugurated may result in a productive industry, all the profits of which shall go to miners, teamsters, smelters or millmen and stockholders, without any deduction for lawyers, judges, juries and experts; and it would be pleasant to see the owners of other land-grants take courage from this example, and deliver forever from the nuisance of extra-lateral rights the mineral lands which they control.

Corundum of the Appalachian Crystalline Belt.

BY J. VOLNEY LEWIS, CHAPEL HILL, N. C.

(Atlanta Meeting, October, 1895.)

CONTENTS.

INTRODUCTION.

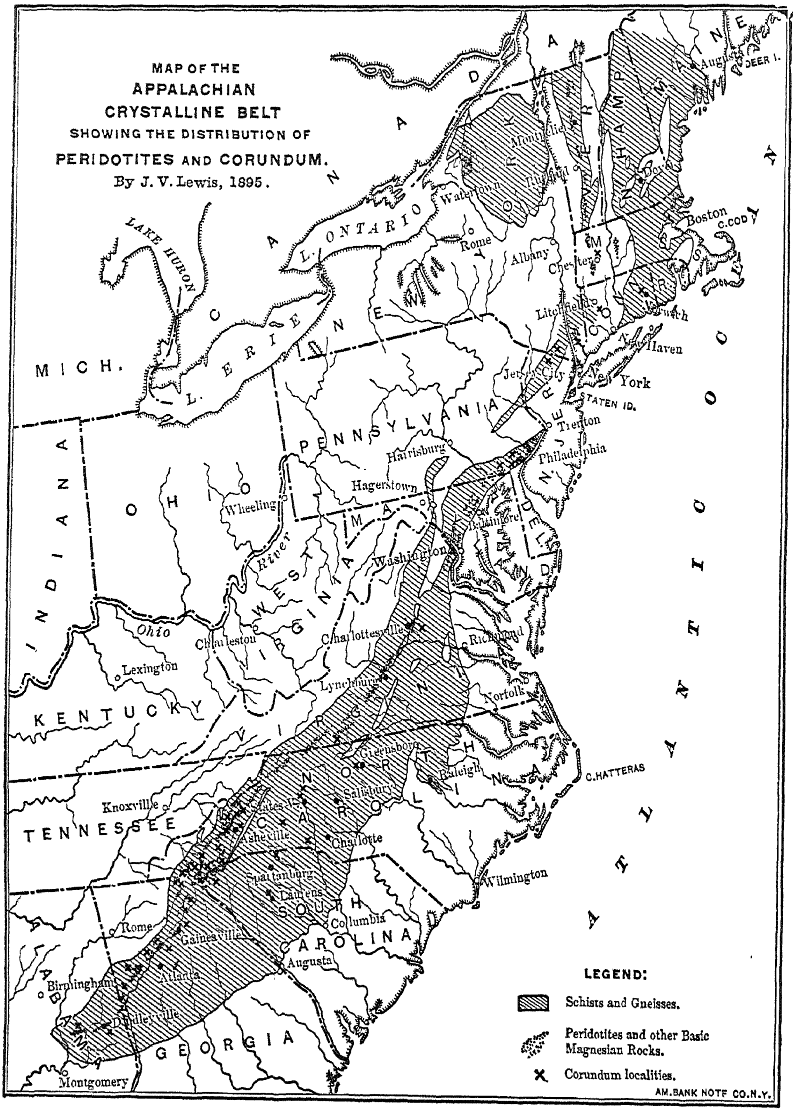
THE following paper is based chiefly on work done for the North Carolina Geological Survey, and is presented here by permission of Professor J. A. Holmes, State Geologist. It represents, in a very general way, the field-results obtained during the summer months of 1893, and 1894, and part of 1895; and the portions pertaining to North Carolina are, to a certain extent, abstracts of a report shortly to be published as *Bulletin* No. 11 of the State Survey. Less emphasis, however, is here laid on the petrographic descriptions of the peridotites, and more detailed treatment is given to subjects pertaining to methods of mining, cleaning, and utilization of the commercial product, corundum.

Some of the most extensive fields for investigation which this subject presents have been scarcely more than suggested.

Among these are—the origin, age, and essential character of the peridotites; the derivation of secondary minerals and rock-types; the seemingly paradoxical conditions that have led to the crystallization of pure alumina with wholly non-aluminous rocks; the subsequent alteration of the corundum itself; the relations of this belt of basic-magnesian rocks to the axes of upheaval of the Appalachian mountain system, etc. These, and other questions which readily suggest themselves to the student of this field, might each become the subject of special investigation and treatment. I have begun the laboratory-study of a series of thin sections with the microscope; and such results as I may be able to obtain on any of the questions suggested, will appear in the publications of the North Carolina Geological Survey. This work has scarcely yet progressed far enough to furnish data for a complete and satisfactory classification of the rocks here considered, and many of the minerals that must be mentioned still await identification by chemical and optical methods. In other words, I must here confine myself mainly to field-observations; and I would urge this consideration as an apology for such errors or uncertainties of nomenclature as may occur in these pages. Theoretical discussions have been studiously avoided, and I have, so far as possible, presented only the facts which may be considered to have been reasonably well determined.

Twenty-five years ago, it was first discovered that corundum is intimately associated with the olivine rocks of western North Carolina. Similar rocks are known in various countries of Europe and other parts of the world; but corundum is nowhere found in such relations outside of the Appalachian region of North America. To this extent, at least, the conditions have been peculiar to this area, and it is only here that we may look for a fuller knowledge of the modes of occurrence of corundum, and a solution of the mysteries of its origin. Since this first discovery, several geologists of note have made hurried excursions to the region, and have published the results of their investigations; but the work thus far done has all been more or less of this nature. None of these writers have known, or attempted to ascertain, the extent of the peridotite belt, or to study its character and variations in widely separated portions of the field. Hence, their studies, though containing work of

considerable value, are local in character, being based on the peculiarities of the particular region visited, and are sometimes misleading in their most important conclusions.



The accompanying map shows the extent of the region under consideration, and the distribution in it of peridotites and corundum.

From an economic, as well as scientific standpoint, it is very

important that comparative studies of the conditions be made in every part of the field. Failures due to ignorance and "boom"-methods have done much to destroy public confidence in Southern mining; and a knowledge of the facts and underlying principles of these deposits, even to a very limited extent, cannot but advance the interests of legitimate industry. It is hoped that earnest, disinterested investigation may be prosecuted in every field of mining, till such disasters are no longer possible. Towards this end, the American Institute of Mining Engineers can do much.

Historical Sketch.

The following sketch is necessarily fragmental, especially as regards the earlier discoveries, since the information upon which it is based has been culled, chiefly, from the mass of literature that has accumulated during the last century. I have not been able, however, to examine the whole field of scientific periodicals, proceedings, reports, etc., in which mention of corundum might be found; but I have collected data from the leading sources, and those most likely to have accurate information on the subject. Among these, I am chiefly indebted to the *American Journal of Science*, the *Mineral Resources of the United States*, and the various State reports.

Besides the references given in the text and foot-notes, a bibliography of the principal literature on the corundum of the Appalachian region is appended to this paper.

Early Discoveries.

The earliest published reference to corundum in America that I have thus far been able to find, bears the date of 1819, and was published in the *American Journal of Science* in 1821. It is a letter from John Dickson, a teacher, of Columbia, South Carolina, who sent to Professor Silliman some minerals that he had collected on a tour through the Carolinas. Among these was an unlabelled specimen of blue corundum—a regular six-sided crystal, three-fourths of an inch long and one inch in diameter, with parting and striæ like the East India corundum. In reply to Professor Silliman's inquiry as to locality, Mr. Dickson writes, "I think it was Laurens district; at all events, it was

picked up by my own hands. . . . . I am sure it is American and Carolinian." *

In 1822 the occurrence of dark grayish-blue corundum in crystals and masses with cyanite at Litchfield, Connecticut, was mentioned by Parker Cleaveland in his *Mineralogy and Geology* (Boston), and the information is credited to J. P. Brace.

In 1827, at a meeting of the Lyceum of Natural History, New York, "Major Delafield . . . . exhibited crystals of sapphire from Newton, Sussex county, New Jersey."† In 1832 Dr. Samuel Fowler described the geologic and mineralogic relations of this corundum, which occurs in the borders of the crystalline limestone adjoining the gneiss.‡

According to Mr. W. W. Jefferis, as quoted by Mr. Joseph Willcox, John and Joel Baily claim to have discovered corundum in the serpentine region of Chester county, Pennsylvania, about 1822 to 1825. Dr. Thomas Seal collected specimens at Unionville about 1832. Mr. Jefferis himself saw large lumps in the fields there in 1837 or 1838,§ but it was not found in place till 1873.

A large detached block of dark blue, laminated corundum was found 3 miles below Marshall, in Madison county, North Carolina, in the spring of 1847. General T. C. Clingman was immediately interested in it, and, after considerable search, a second piece was found in 1848.|| This was more than twenty years before the discovery at Corundum Hill, and the locality, so far as I can learn, has never furnished another specimen. In the next year, 1849, Dr. J. L. Smith announced the discovery of emery *in situ* in Asia Minor.

In 1852 Mr. W. P. Blake described corundum from Vernon, Sussex county, New Jersey;¶ and in the same year Dr. C. L. Hunter found "emery" and corundum in place in Gaston county, North Carolina.**

In 1864 Dr. C. T. Jackson predicted the occurrence of emery at Chester, Massachusetts, from the discovery of margarite, a

* John Dickson, *Am. Jour. Sci.*, 1, iii., 1821, pages 4, 229, 230.

† *Am. Jour. Sci.*, 1, xiii., 1828, page 380.

‡ Samuel Fowler, *Am. Jour. Sci.*, 1, xxi., 1832, pages 319, 320.

§ Joseph Willcox, *Second Geol. Survey of Penn.*, C 4, 1883, pages 346 to 351.

|| *Report N. C. Geol. Survey*, i., 1875, appendix C, page 64.

¶ W. P. Blake, *Am. Jour. Sci.*, 2, xiii., 1852, page 116.

** C. L. Hunter, *Am. Jour. Sci.*, 2, xv., 1853, page 376.

mineral which Dr. Smith had just reported as characteristic of the emery-deposits of Asia Minor. In September of the same year the emery was found by Dr. H. S. Lucas.* The deposits had previously been thought to be only magnetite, and some of it had actually been smelted along with other ores. Two years later, distinct corundum crystals were also found there.† This discovery of emery soon led to the establishment of actual mining, the first of its kind in America. The mine is still operated by the Hampden Emery and Corundum Company, though it has not been worked continuously from the beginning. The same company owns the corundum mines at Corundum Hill and Buck creek, North Carolina, and that at Laurel creek, Georgia.

Up to this time, though several localities of mineralogic interest had been discovered in North Carolina, the principal mining regions of to-day were still unknown. In 1870 Mr. Hiram Crisp was living on what is now known as Corundum Hill. His attention was attracted by peculiar rocks which were scattered over the surface, and he carried specimens to Major Higdon, who, in turn, took them to Raleigh for the inspection of Prof. W. C. Kerr, then State Geologist. The discovery that these masses were corundum aroused considerable interest, and General Clingman, Dr. C. D. Smith and others soon instituted in the region a search which resulted in bringing to light several other localities.

In 1871 Mr. J. H. Adams described the occurrence of corundum in vermiculites, with asbestos and other amphiboles, at Pelham, Massachusetts.‡ Prof. B. K. Emerson, of Amherst College, writes me that this occurrence is associated with olivine rocks, and is very similar to those of the south Appalachians. So far as I am aware, this is the only instance of the kind north of North Carolina.

In the seventies, after the Corundum Hill discovery, great activity prevailed throughout this region in the search for new localities, chiefly, however, with a view to finding gems. This soon resulted in bringing to light numerous occurrences with the peridotite of the adjoining counties of Clay, Macon, Jack-

* C. T. Jackson, *Am. Jour. Sci.*, 2, xxxix., 1865, pages 87 to 90.

† C. T. Jackson, *Am. Jour. Sci.*, 2, xlii., 1866, page 421.

‡ J. H. Adams, *Am. Jour. Sci.*, 2, xlix., 1870, pages 271, 272.

son, Transylvania, etc. Prospecting was begun on the Corundum Hill property by Dr. C. D. Smith and Mr. Crisp, and about a thousand pounds of crystals and lumps were taken out near the surface, some of them weighing as much as forty pounds.* In the fall of 1871 this property was purchased by Colonel C. W. Jenks, of Boston, and Mr. E. B. Ward, of Detroit, and mining was soon begun under the management of Colonel Jenks.†

In the same year Dr. F. A. Genth, while investigating some iron ores in the central portion of North Carolina for the State Geological Survey, discovered emery in the magnetite belt of Guilford county.‡ Its extent was not determined, as the specimens were not recognized till they were examined in the laboratory.

In 1874 or 1875 Mr. J. A. D. Stephenson found corundum on the surface in Iredell county, and it has since been discovered in place with amphibolites, and also found in a few places in the adjoining county, Alexander.

From that time to the present the search for corundum has not ceased throughout all this region, though it has been prosecuted with varying degrees of activity, as waves of excitement have risen and slowly subsided. The result has been to extend greatly the known limits of its distribution; but, as yet, no lasting operations have been established except the mines at Corundum Hill and Laurel creek. In the aggregate, considerable capital has been invested in other property, and much has been expended in unsystematic prospecting which could not be expected to accomplish anything. But dearly-bought experience is beginning to exercise a salutary influence as regards both investing and prospecting; and indications are now more favorable than ever before for a healthy, natural development of the corundum belt.

Principal Mines.

The following are brief sketches of all workings that may, in any proper sense, be termed mines. Of these, the Corundum Hill, the Sapphire, and the Laurel Creek mines, are, at present, by far the most important. Extensive prospecting has been

† *Geol. Survey of N. C.*, i., 1875, appendix, page 103.

‡ See Dr. Raymond's paper on "The Jenks Corundum Mine," *Trans.*, vii., 83.

§ *Report N. C. Geol. Survey*, i., 1875, page 246.

done at a number of places which have come to be known locally as "mines," and considerable quantities of corundum have been taken from some of these, but none of them have yet entered the market with their products. Machinery is now being installed however, for the immediate operation of mines on Higdon Knob, Macon county, North Carolina, and on the west slope of Chunky Gal mountain in Clay county.

Portions of the historical data for these sketches have been gathered piece-meal through the country from men who have been employed in the work; but, in regard to the more prominent mines, I am chiefly indebted to the superintendents and owners for most of the facts here presented. I would especially acknowledge the kindness of Dr. H. S. Lucas, who first placed corundum mining on a successful basis, and who, for twenty years, has been identified with the industry; and of Mr. Charles N. Jenks, who was associated with his father, Col. C. W. Jenks, in the first corundum mining ever undertaken, and who has been, in later times, actively engaged in the management of the Sapphire mines. Valuable information has also been furnished by Mr. L. S. Ropes, Dr. C. Grimshawe, and Mr. A. M. Stoner. Where assistance has been derived from publications, references are given in the text.

Sketches are here given of the following mines: In Georgia, the Laurel Creek, and the Track Rock; in North Carolina, the Corundum Hill, Behr, Buck Creek, Sapphire, Carter, and Acme; in Pennsylvania, the Unionville mine.

Laurel Creek Mine, Pine Mountain, Rabun County, Georgia.—The first prospecting at this locality was done for asbestos, and in this work corundum was accidentally found in the early seventies, by an Englishman named Thomson. Col. C. W. Jenks, who had already begun work at Corundum Hill, bought the property and began prospecting in the spring of 1872. Work of this nature was continued, at intervals, for two or three years; but, though some fine specimens were secured, corundum was not found in large quantity. In 1880, regular mining was begun by Dr. H. S. Lucas, who had successfully reopened the mine at Corundum Hill. Mills for crushing and cleaning were erected on the creek, and for thirteen years the mine was successfully worked, yielding about 300 tons of corundum a year.

By the necessity of location, the principal deposit, near the south end of the peridotite mass, had to be worked by a shaft. At the time of closing down, in 1893, this work had reached a depth of 140 feet, most of it below the level of the creek; and a pump with a 5-inch discharge-pipe was barely sufficient to keep the mine free from water. No work has been done since 1893; but I am informed by the president of the company that plans are on foot for reopening the mine, though this course has not been definitely determined.

Here, as at Corundum Hill, the mills erected at the mine were only for the purpose of cleaning. The corundum was hauled on wagons to Walhalla, South Carolina, a distance of twenty miles, and shipped to the mills at Chester, Massachusetts, for final preparation for the market.

*Track Rock Mine, Track Rock Gap, Union County, Georgia.**— This mine was equipped with a steam cleaning-plant, and operated by the New York Corundum and Mining Company for a short time in 1893. Several tons of corundum were shipped, but no work has been done since that date. The corundum occurs in an amphibolite, which, on the surface, is considerably altered to chlorite. The irregular fragments of corundum are wrapped in a coating of pearly margarite. In portions of the rock the microscope revealed the presence of olivine, and the position of the outcrop is in line with the peridotite, which appears at short intervals in the valleys of Brasstown and Shooting creeks to the north. Mining was confined to a drift of about 200 feet on the south side of the gap.

Corundum Hill Mine, Cullasaja, Macon County, North Carolina. —This mine is noted not only for being the pioneer of its kind, but also for having proven the most successful yet undertaken. The first discovery of corundum here, and the beginning of operations by Col. C. W. Jenks, in 1871, have already been noted in the historical sketch above. The first mining was conducted chiefly for gems and fine cabinet specimens, and much valuable "sand-corundum" was allowed to go to waste. In fact, the possibility of mining corundum on a large scale for abrasive purposes had hardly yet been conceived. On account of the scarcity of gems, the work was soon found unprofitable,

* For information in regard to this mine I am indebted to *Bulletin* No. 2, of the Geological Survey of Georgia, 1894.

and the mine was closed down in 1874. A number of fair gems had been obtained, and about a hundred tons of lump and crystal corundum were sold for cabinet specimens and for the manufacture of dental tools.

In 1878, the mine was bought for the Hampden Emery Company, of Chester, Massachusetts, by Dr. H. S. Lucas; and, during the same year, work was resumed and has continued without interruption to the present time. From 200 to 300 tons of cleaned corundum a year have been mined, though the output was somewhat smaller during 1893–94, owing to the depression in iron manufacturing and other related industries. The work has been chiefly confined to two large open cuts, and a series of drifts beneath one of these. At present, all work is done in drifts, and one of these penetrates the hill to a distance of over 600 feet along the southeastern border of the peridotite. Mills for cleaning are erected at the mine, but the material is hauled twenty miles to the railroad and shipped to Chester, Massachusetts, for sorting into sizes and marketing.

Behr Mine, Elf, Clay County, North Carolina.—This mine is on Shooting creek, about 5 miles east of Hayesville. A little work was done here in 1880 by Dr. H. S. Lucas. Some time afterwards the mine was bought by Hermann Behr & Co., of New York, and operations were begun on a larger scale. A steam cleaning-plant was erected, and considerable work was done; but in answer to inquiries as to the output, I could only learn that "several car loads were shipped." The location is not favorable to economic mining; and much of the work must have been of the nature of prospecting. The mine is situated in a low place beside a branch, so that pumps were constantly required to keep it free from water. The nearest railway shipping points are 35 or 40 miles away, and transportation by wagons over this distance must constitute a great obstacle to the reopening and further development of the mine.

Buck Creek (Cullakanee) Mine, Clay County, North Carolina.—In the *Report of the North Carolina Geological Survey* for 1875, Dr. C. D. Smith says that he was the first to find corundum at Buck creek. Large loose blocks with feldspar and black hornblende were lying on the surface. In regard to the work done here, I have found it possible to get only fragmentary information.

About 1875 the locality was prospected by Major Bryson;

two years later Mr. Frank Meminger worked for six months, and is said to have taken out about 30 tons of corundum. For a period of ten years no further operations were undertaken: then Mr. Ernst took charge of the place, and conducted prospecting for nine months. During another period of four years the mine was idle. Then work was undertaken on a liberal scale by Mr. Gregory Hart, and continued for a year and a half, during which time a shaft was sunk on the feldspar vein, and several open cuts were made in the chlorite zones. Though considerable quantities of corundum were taken out during this work, it was chiefly massive "block-corundum," and was left on the ground for want of economic means of crushing and cleaning it. In 1893 the mine was purchased by the Hampden Emery and Corundum Company (as it is now styled), and the corundum already mined was cleaned at the mills at Corundum Hill and shipped. Since then a little work has been done in the chlorite zones, but no further mining has been undertaken.

It is 40 miles to the nearest available railroad station, and three considerable mountains are crossed on the way. But, by the construction of only a few miles of road, another station could be reached within 20 miles, with only one mountain to cross.

Sapphire Mines, Sapphire, Jackson County, North Carolina.— The cleaning-mills of this property are very near the Jackson-Transylvania county-line, and workings are located in both counties, five in Jackson and three in Transylvania. All these are clustered about Great Hogback mountain, and are sometimes referred to as the "Hogback mines." Dr. C. D. Smith, in his report on the corundum region, in 1875, speaks of several hundred pounds having been obtained in this vicinity by digging small pits. No mining was done, however, until 1892, when the Sapphire Valley Company began work at the "Burnt Rock" mine, in Transylvania county, 7 miles northeast of Sapphire. A number of other places were soon opened in the vicinity, and a complete crushing- and cleaning-plant was erected. The country was almost an uninhabited forest region; and great expense was incurred in building roads, bridges, houses, shops, etc. For about a year fifty or sixty men were employed in mining, prospecting, and improvements; but operations were suspended during the financial panic of 1893, and have not since been resumed. Mr. Charles N. Jenks, under whose

management the work was done, and to whom I am indebted for this sketch, informs me that the product during the year of activity was over 400 tons, one-fourth of which was crystal corundum, 90 per cent. pure.

The cleaned corundum was hauled on wagons to Hendersonville, a distance of some 40 miles. Since the completion of the railroad to Brevard, a considerable saving will be effected in transportation. The property has recently been purchased by Mr. C. H. Stolzenbach, of Pittsburgh, Pa., but the management of the mines will continue in the hands of Mr. Jenks.

Carter Mine, Madison County, North Carolina.—This mine is located at the northern extremity of a long peridotite outcrop, which lies chiefly in Buncombe county. Corundum was first found here by Dr. C. D. Smith, fifteen or twenty years ago. The first prospecting was done by Mr. William Carter and Dr. H. S. Lucas, and similar work was performed at intervals afterwards by Mr. M. E. Carter, and subsequently by Messrs. Coleman and Rice. Regular mining was begun in 1886 by Tarr, Hamilton & Co., who built a steam-plant for crushing and cleaning. After about six months' work and the production of about 20 tons of cleaned corundum, the mining was stopped, and has not since been resumed.

Acme Mine, Statesville, Iredell County, North Carolina.—About 1875 Mr. J. A. D. Stephenson found corundum near the site of the present operations, about three-fourths of a mile west of Statesville, and half a mile south of the Charlotte and Taylorsville railroad. Surface-specimens were found and some prospecting was done at a number of localities in the region for 15 miles north and west of Statesville. But the sources of these specimens were found in only two places; namely, at the location of the mine under consideration, and 7 miles west, where corundum occurs in vermiculite and feldspar through a dark green hornblende-rock, as described below under "Modes of Occurrence."

The Acme Corundum and Mining Company began operations in February, 1893, under the management of Mr. H. A. Collins. The first product consisted of large rough blocks and crystals, which were sold just as they came from the mine. A steam-mill was soon erected, however, and the product of cleaned material in 1893 was about 50 tons.

The mine is situated in a depression near the head of a branch, where the alluvial clay-deposits are about 15 feet thick. The decomposed hornblende-rock underlying this clay is also very soft and saturated with water, and great difficulty was encountered in keeping these materials out of the workings. The source of the corundum was a feldspar-vein traversing a zone of the vermiculite in the hornblende-rock. During 1894 no work was done until December. Prospecting was then begun for the purpose of locating deposits under more favorable conditions.

*Unionville Mine, Newlin Township, Chester County, Pennsylvania.**—As already pointed out in the historical sketch above, corundum is said to have first been found here before 1825, and specimens were collected on the surface for a number of years before any work was attempted. Large blocks of corundum were scattered over the ground, one of them weighing, according to Mr. Jefferis,† over 5000 pounds. Many such masses are said to have been under-dug by the farmers and buried to get them out of the way. A few tons of lumps and crystals are said to have been collected and shipped to London about 1839. One of the sources of this floating corundum was found in 1873 in what appeared to be a large bed or vein of corundum in margarite, with a little diaspore. It soon proved, however, to be only a great lenticular mass, and was quickly exhausted. Work was continued for many years, but, with the exception of this mass, the output was at no time large. Deposits were mined both in the feldspathic gangue and in the corundum-bearing chlorite. The principal shaft was sunk to the depth of 150 feet, and three or four years ago a drift was put in to drain the working, and with the hope that in its course corundum would be found. It penetrated to the shafts, but was not successful in finding corundum in paying quantities, and mining has ceased.

Geology of the Corundum Region.

What I have called the *peridotite belt* is included in a broader area of crystalline rocks, which is coextensive with the Appala-

* I am indebted to Mr. Theodore D. Rand, of Philadelphia, for valuable information in regard to this mine.

† *Proc. Acad. Nat. Sci.*, Phila., 1892, page 188.

chian mountain-system, and which, on account of its complex and highly crystalline character, is generally considered to be of Archean age. This crystalline belt appears in central Alabama, where it emerges from beneath the coastal plain formations, and is continuous to New Jersey, where it disappears for a short distance beneath the Newark (or Jura-Trias) at Trenton. Northeastward, the crystalline rocks form irregular areas in northern New Jersey, New York and Connecticut, and the other New England States.

The principal constituent of these rocks is gneiss, often passing into schists, however, and containing frequent masses of granite and dikes of more basic igneous rocks. The gneisses are usually considered to be, in great part, sedimentary rocks which have lost their original clastic characteristics (with the possible exception, in some cases, of bedding) in the great earth-movements and other metamorphosing processes in which they have been involved. Some of these gneissic rocks, however, are found to be simply sheared igneous rocks, especially granites, and the transition may be seen about the borders of many of the granite masses. When the shearing action has not gone too far in the destruction of the original structure, these transitions may be seen in the field; and, even when not observable in this manner, the microscope still reveals the typical granitic structure and sequence of crystallization.

In much of the work of earlier geologists in gneiss-areas, the planes of lamination were regarded as stratification, and the attempts to interpret structure were based on this conception. It is now, however, a generally recognized principle that, in the process of metamorphism, such planes may be produced in clastic rocks at any angle with the original bedding; and the more thorough the metamorphism in such rocks, the less probability there is of any recognizable trace of the original structure being retained. This fact, coupled with the mechanical production of gneissic lamination in massive rocks, renders the question of structure in such regions extremely complex.

Strikes and dips, when noted in the following pages, invariably refer to the planes of lamination. In North Carolina these planes have a prevailing strike of about north 30° east, and usually dip southeast at a high angle. But the dip is subject to considerable variation both in angle and direction, often pass-

ing through the vertical from one side to the other, and this sometimes occurs in a space of only a few feet. In some localities, especially in the immediate vicinity of massive rocks, the gneisses have been twisted into the most gnarled and contorted forms imaginable. In such places, also, the gneiss often passes locally into mica-schist, though this development is by no means confined to such areas.

Occupying a narrow belt about the middle of this crystalline region are found the basic magnesian rocks, which concern us here chiefly in their relation to the corundum-deposits. They form disconnected lenticular masses, in a line or series of parallel lines, which usually coincide with the general direction of the strike of the gneisses. In respect to size, these masses are, as a rule, small, ranging in width from a few feet to a few hundred feet, and having an average of less than a thousand feet in their greatest dimensions. The lenticular form is quite characteristic, but it is often drawn out, and sometimes gives place to an outcrop of uniform width for distances of 2 or 3 miles. Further northward, as in Maryland and Pennsylvania, where the peridotites exist only in their hydrated form (serpentine), these greater dimensions seem to be less exceptional. But many of the serpentine areas that we are accustomed to see represented on the maps as continuous masses, are resolved, on examination in the field, into a series of small lens-shaped bodies, and these bodies are entirely similar in form, position and associated minerals to the unaltered peridotites of North Carolina and Georgia.

As to the nature of these rocks, geologists are not by any means agreed. Whether they are deposits formed by crystallization from cooling solutions, as Dr. Hunt maintained; or olivine sand derived from a pre-existing volcanic rock, as Professor Julian concluded; or whether we may regard them as igneous rocks intruded into the gneisses as laccolites or dikes, as Wadsworth and others believe; or, finally, whether they may not be derived from ancient dolomitic limestones, as Rosenbusch, in later years, seems to think—remains to be decided by further investigation. When local occurrences only are considered, evidence may be gathered that seems to sustain one or another of these theories, according to the locality chosen and the personal equation of the observer.

A discussion of the question of the origin of the peridotite would be out of place in this connection; besides, the material which I have on this subject is not yet in form for presentation. By the work which I have begun, however, I hope to obtain in the near future some results worthy of publication.

In North Carolina, besides the crystalline rocks mentioned, two narrow belts of sedimentary rocks are found. The broader of these lies on the western border of the State, between the gneisses of North Carolina and the Paleozoic rocks of Tennessee; the other, quite a narrow strip, is situated about 30 miles further east, and both lie approximately parallel to the trend of the gneisses. This formation, according to Mr. Arthur Keith, of the U. S. Geological Survey, consists of a lower series of shales and limestones, followed by sandstones and conglomerates, and the whole lies unconformably on the gneisses. This series has been called *Ocoee;* but as no fossils have been found in it, and as no correlation with known formations is yet possible, its position in the geologic column is unknown. The peridotite is scattered over the area between these two strips of Ocoee, and is found in some places very near their borders; but, as yet, no rocks of this or similar character have been found *within* the Ocoee.

Rocks of the Peridotite Belt.

As the chief corundum-deposits are found in connection with peridotite, it is important, before passing to the consideration of the corundum, to describe briefly the various types of rocks represented. These may be classed in four groups, namely, peridotites, pyroxenites, amphibolites and secondary rocks. Of these the peridotites largely predominate, and the other groups indicate only variant or accompanying forms, which, however, sometimes attain important development as independent rock-masses. Under the group of secondary rocks are included chiefly the hydrated forms of the other groups.

Peridotites.

The typical lens-shape of these outcrops has been already mentioned, and also the long-drawn-out modification of this form; but still other variations—such as the sending-off of branches into the surrounding gneiss, or the presentation of a

forked outline—are seen in a number of places, and numerous irregularities are found in individual instances. In most of the outcrops the rocks are perfectly massive and devoid of all regularity of structure, except a parallel development about the borders; and sometimes this is absent. But there are also a number of places in which lamination is so highly developed as to give the peridotite a close resemblance to a thin-bedded sandstone. The exposure at Webster, North Carolina, is the most prominent example of this kind.

No true contacts between the gneiss and peridotite are to be found. There always intervenes a zone of secondary hydrous minerals, bearing also more or less alumina; and it is in these border-zones and their extensions into the joints of the massive rocks that the corundum is found. These so-called veins are composed of enstatite or talc, forming a radiating sheath, arranged perpendicularly to the surface of the peridotite, and just outside of this is a belt of chlorite and vermiculite, very variable in thickness and often bearing corundum. The peridotite is always more or less altered where these zones are highly developed, and often within the enstatite casing it is completely decomposed, the enstatite also being altered to talc. The least altered specimens of peridotite are found where little or no chlorite is developed. When completely decomposed, the space enclosed by the enstatite casing is filled with a soft mass of ocher, and the residual silica is deposited along the joints as chalcedony.

The gneiss in contact with these zones has usually the appearance of the normal, unaltered rock; but on handling it is found to be extremely friable at the surface, crumbling to a mass of sand almost at a touch. This condition usually extends for several feet from the border, where it again regains its natural firmness. These border-zones are planes of considerable movement, as is shown by the slickensides in the chlorite and especially along the talc-covered face of the peridotite. This shearing action has, doubtless, some intimate connection with the formation of the series of intermediate minerals as well as the condition of the adjoining rocks on either side.

Even assuming an igneous origin for the peridotite, the masses are too small to have produced any appreciable contact-action on the adjoining gneisses, and we should hardly expect to find

any trace of it now in the presence of so much evidence of subsequent change. Besides, the more basic rocks are known to have very little contact-effect, and hence we should not expect it in this case, the peridotite being at the bottom of the basic series.

Brief mention will now be made of the more important rock-divisions represented in the peridotite belt; but it must be borne in mind that these divisions are chiefly mere mineralogic varieties, and correspond to no well-defined lines in nature. In fact, specimens from different portions of the same rock-mass would often show two or three entirely distinct types, according to the classification generally received, and in the field it is readily seen that the peridotites are a petrographic unit. With this point clearly in mind, the terms commonly used to designate the different types may be serviceable for convenience and closer accuracy; but the distinctions must not be pressed to too great a refinement.

The peridotite may be subdivided into the following varieties: *dunite*, the rock composed essentially of granular olivine, with occasional grains of picotite or chromite; *harzburgite*, the olivine rock with the addition of enstatite; *amphibole-picrite*, the olivine-hornblende variety; and *forellenstein*, composed of olivine and feldspar. According to the strict definition of peridotite—an olivine-bearing rock without feldspar—the type last mentioned cannot be considered to belong to this group; but here they are undoubtedly phases of the same rock, and we cannot consistently separate them.

Dunite.—The Appalachian phase of this rock is quite similar to that of the type-locality in New Zealand—a yellowish to grayish green, crystalline-granular rock, with a glassy to oily luster and an irregular fracture. Besides the granular olivine, the grains and octahedra of picotite (or chromite) constitute a characteristic accessory. By their coffee-brown translucence under the microscope, these grains sometimes appear to be picotite, while in other specimens, or even sometimes in other portions of the same grain, the mineral is perfectly opaque. Until further investigation, I have provisionally called the translucent variety picotite and the opaque chromite, though they sometimes grade insensibly into each other.

A light oil-green seems to indicate the freshest condition of

the rock, while yellow and brown colors usually indicate partial decomposition, and the dark green is due to the formation of serpentine. The microscope shows nearly all available specimens to be more or less altered, and in this process a number of secondary minerals are formed. The most common is a network of serpentine about the olivine; needles of tremolite often penetrate the olivine grains, and flakes of talc are frequently seen.

The characteristic dull brown color of the weathered surface, which gave the name to the type-locality and to the rock itself, is seen in nearly every outcrop. An ocherous soil is the product of final decomposition, but it supports very little vegetation and is readily washed away, leaving a bare, rocky surface. Blocks of massive or honey-combed chalcedony and fragments of talc are almost universally present.

Harzburgite (Saxonite).—This rock is the same as the dunite described above, except that enstatite has assumed important macroscopic proportions, sometimes becoming locally the predominant mineral. The enstatite often has the bronzy sheen which usually distinguishes the variety bronzite. Frequently, in surface-exposures, this mineral has been altered entirely into talc, while still retaining the brilliant luster and bronzy sheen of the unchanged mineral. By the suppression of olivine we arrive at the pyroxenite described below as enstatite-rock, and the alteration of this type produces most, if not all, of the talc-rocks of the region. Typical North Carolina localities for harzburgite are Balsam gap, Jackson county; Elk creek, Ashe county; Woody Place, Mitchell county, and Ray's chrome-mine, Yancey county.

A talcose rock bearing large crystals of olivine, often twinned in the form of a cross, is found near Philadelphia in a belt stretching from near Bryn Mawr to Chestnut Hill. This rock was named by Dr. H. Carvill Lewis *peridosteatite.* Outcrops of a very similar nature occur northwest of Marshall, in Madison county, and in Ashe and Alleghany counties, North Carolina, and it is highly probable that they are derived from the partial alteration of harzburgite. This rock is practically identical with the *glinkite* of the Urals.

Amphibole-Picrite.—This is also identical with dunite, except for the presence of a hornblende. In the places which I have

observed it was a light green hornblende, with the optical properties of actinolite. Instead of weathering to talc, however, this mineral produces chlorite, and many of the chloritic peridotites are doubtless derived from this type. Suppression of the olivine produces amphibolite, which is, undoubtedly, the origin of many of the chlorite rocks. The chief development of this type is found in the northern portion of the great peridotite mass at Buck creek, Clay county, North Carolina.

Forellenstein (Troctolite).—This rock consists essentially of olivine and feldspar (anorthite), and the zones of reaction-products which always appear between these two constituents. These minerals have been described minutely by Dr. F. D. Adams, who regards them, on the ground of optical investigation, as hornblende and enstatite. They appear as a double zone radiating perpendicularly to the surfaces of the minerals enclosed, the hornblende zone being next to the olivine, with the enstatite on the border of the feldspar. Several localities have this type developed, but it is seen best at the Buck creek locality referred to above.

Pyroxenites.

Two types of pyroxenic rocks are found in this belt, both in closest relations with peridotite. The more abundant is that composed entirely of orthorhombic pyroxene, *enstatite-rock*, which is widely distributed, and frequently forms with peridotite portions of the same rock-mass. The other type is that consisting of both orthorhombic and monoclinic pyroxenes, and was named *websterite* by the late Dr. George H. Williams, from its type-locality at Webster, North Carolina.

Enstatite-rock.—The essential constituent of this rock is usually in large, interlocking, bladed crystals of a grayish or yellowish gray color. It is typically developed at Laurel creek (Pine mountain) corundum mine, in Rabun county, Georgia, and at Corundum Hill, in Macon county, North Carolina. In both these places it forms considerable masses in perfect continuity with the peridotite, though the proportion is much greater at Pine mountain. In many places it is present in varying proportions with peridotites, and especially with those bearing enstatite. In a number of localities in Georgia and North Carolina it also forms apparently independent masses, though some of these have been prospected for corundum and found

to contain more or less peridotite below the surface. Evidence has been found which points to the derivation of this pyroxenite from the olivine-rocks; but we cannot discuss this question here.

Many of the masses have been entirely altered into talc on the surface, and all of them show more or less of this tendency. Even when completely altered the rock usually retains the structure and appearance of the original, unless it has also been considerably sheared. Investigation will probably show that many of the talcose rocks of Virginia, Maryland and Pennsylvania have had a similar origin. Some of the serpentines of Pennsylvania are said also to bear evidence of having been derived from enstatite-rock.

In view of the prevailing tendency to apply the name anthophyllite to these rocks,* I have had analyzed by Dr. Charles Baskerville, of the University of North Carolina, a typical specimen from Corundum Hill, with the result given below in column I. The high percentage of water shows considerable alteration, but the result is unmistakable. Eliminating the 5.45 per cent. of water and calculating the remaining constituents on the basis of 100, we obtain the results given in column II., and these figures represent a normal enstatite, with an iron percentage which places it near the bronzite variety. For comparison, an analysis of another specimen from Corundum Hill, made in the laboratory of the University of Pennsylvania by Mr. Frank Julian† is given in column III.

Analysis of Enstatite from Corundum Hill, Macon County, North Carolina.

	I.	II.	III.
SiO_2,	51.64	54.95	57.30
Al_2O_3,	0.12	......	trace
FeO,	9.28	9.87	7.45
CaO,	0.45	......	......
MgO,	31.93	33.97	34.64
MnO,	0.56	......	......
H_2O,	5.45	......	1.21
Totals,	99.43		100.60

Websterite.—This rock, composed of the two classes of pyrox-

* Dr. C. D. Smith, *Report North Carolina Geological Survey*, I., 1875, Appendix, page 93. Francis P. King, *Bulletin* 2, Geological Survey of Georgia, pages 79, 82, etc.

† *Bulletin* 74, United States Geological Survey, "Minerals of North Carolina," 1891, page 43.

ene in varying proportions, is found, as far as I am aware, only at the type locality, at Webster, North Carolina, and on Cane creek, about 6 miles east of that place. It is a compact, granular rock, closely resembling dunite, which it was considered to be until its true character was recognized by Dr. Williams. It forms a mass about 300 feet wide, and recognizable for nearly a mile in the great body of laminated dunite at Webster. It is more massive in character than the dunite, has a more brilliant green color, and its surface is less affected by weathering. It supports a vigorous forest-growth of oak and hickory, which is flanked on either side by almost barren dunite.

Amphibolites.

This term is provisionally adopted to include several types of rock, bearing different species of amphibole, and often considerable feldspar. One of the most important of these is the feldspathic rock, bearing beautiful grass-green amphibole, commonly called "smaragdite." The feldspar is anorthite and the proportion between the constituents varies all the way from a nearly pure hornblende-rock to that composed only of anorthite. It is usually gneissic in structure, but is also found massive. In Clay county, North Carolina, and Towns county, Georgia, it bears a considerable proportion of laminated pink and ruby corundum, and the combination of these brilliant colors forms exquisite cabinet-specimens. The corundum varies in size from the minutest grains, visible only in thin sections under the microscope, to occasional plates three or four inches in width. Blocks gathered up from the surface at Buck creek have been hauled to Corundum Hill and crushed for the separation of the corundum; but as the rock is exceedingly tough and the proportion of corundum is very variable, it is not likely to become a commercial source for this mineral.

The amphibole constituent, on account of its bright grass-green color, has always been called *smaragdite.* Dr. Genth found it to be an aluminous mineral, and doubted the accuracy of this usage, but still retained the name.* On the basis of Dr. Genth's analysis, Prof. Dana placed it in the species *edenite.*

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* *Bulletin* 74, U. S. Geological Survey, 1891, page 45.

Under the microscope, the more brilliantly colored grains are all found to contain minute inclusions of picotite, and the color of these grains is undoubtedly due to the presence of chromium. This will explain the higher specific gravity and chrome-percentage found by Dr. Genth, who used the grass-green variety. I have had the same mineral analyzed, after separating in the Thoulet solution the grains with the greatest amount of picotite inclusions, with the result given in column I. below. Dr. Genth's analysis is given in column II.:

Analyses of Aluminous Hornblende from Buck Creek, Clay County, North Carolina.

	I.	II.
SiO_2,	44.38	45.14
Al_2O_3,	17.32	17.59
Cr_2O_3,	0.38	0.79
FeO,	3.83	3.45
NiO,	......	0.21
MgO,	15.48	16.69
MnO,	0.90	......
CaO,	11.51	12.51
Na_2O,	1.24	2.25
K_2O,	0.38	0.36
H_2O,	4.63	1.34
Totals,	100.05	100.33
Specific gravity,	3.075	3.120

Another amphibole-rock, without feldspar or corundum in the body of the rock, is found in the corundum-localities near Statesville, North Carolina, in the same relation as the peridotite of the mountain region. That is, it forms a massive rock in the gneiss, with corundum-bearing zones of vermiculite and chlorite about the borders and in the joints. The rock is somewhat coarse, and composed of interlocking bladed crystals of dark green hornblende, and often with scales of brown vermiculite. A similar amphibolite is found at the Presley mine, on the north fork of Hominy creek, Haywood county. Through this rock are numerous veins of coarse pegmatite, bearing corundum, surrounded both by mica and feldspar, and sometimes coated with margarite. The amphibole of the Track Rock mine in Union county, Georgia, seems to be chiefly actinolite, with occasional olivine, from which it is probably derived.*

* *Bulletin* 2, Geol. Survey of Georgia, 1894, pages 92 to 95.

Secondary Rocks.

The most important member of this group is *serpentine*, which, in many places in this belt, is but a hydrated form of peridotite, though the serpentines of Pennsylvania, as already mentioned, are said to bear evidence of having been in part derived from pyroxenite. In either case, they must be considered as belonging to this belt. Besides the serpentine, we also have, in considerable development, the talc and chlorite rocks.

Serpentine.—The few thin sections I have seen of the Pennsylvania and Maryland serpentine show quite a massive rock, with little or nothing to give a clue to its origin. No fresh olivine-rocks are found in connection with them, or in fact anywhere throughout this portion of the belt, except an olivine-bearing pyroxenite in Pennsylvania; but the presence of talc and chlorite flakes and grains of chromite through them give them a striking similarity to the peridotites of Georgia and North Carolina; and the resemblance is still more impressive in the border-zones of corundum-bearing chlorite found in Pennsylvania, and the great accumulations of chromite in the mines of both Pennsylvania and Maryland. No Virginia localities have been visited, except where the peridotite crosses the North Carolina line, just west of the Blue Ridge. Serpentine is known, however, in a number of localities along the direction of this belt, where it has been presumably derived from peridotite. In North Carolina the olivine-rocks are more or less serpentinized over a considerable area in portions of Buncombe and Madison counties. In every instance, however, the origin is easily seen in thin sections under the microscope, and sometimes the olivine remnants are plainly visible to the unaided eye.

Talc.—The frequent alteration of enstatite-rock to talc has been pointed out in the description of that rock. By far the greater portion of the talcose rocks of the peridotite belt has been produced by this means, and probably all of it has been formed by the production of the mineral enstatite as an intermediate step.

Throughout the region traversed by this belt, the soapstone thus derived is utilized in the construction of fire-places and for tombstones. In the northwest corner of North Carolina the quarrying of soapstone for furnace-linings became quite an industry while the copper-mines were in operation at Ore Knob

and at Elk creek, and extensive quarries are worked in Pennsylvania and Maryland.

About the borders, and frequently along the joints, of the peridotite, more or less talc is always found; but this is usually from the alteration of the radial enstatite-fringe found between the peridotite and the chlorite-zone. Such a fringe is sometimes very slightly developed, and it is often completely altered and reduced by shearing to a thin layer of schistose talc. The talc of these border-developments never attains the importance of a rock, though the constituents of these borders may sometimes be found for a considerable distance beyond the peridotite, forming a thin strip in continuation of the greater axis of the lens-shaped mass. These strips sometimes connect otherwise independent masses over distances of 2 or 3 miles. In the intervals, the talc rarely shows a development of more than a few feet in width.

Chlorite.—There is always a zone of chlorite of very variable thickness where corundum occurs about the borders of the peridotite, and in many places where corundum is not found. But this, like the accompanying talc, never acquires sufficient development to be considered as a separate rock. Much of the peridotite on the north of the great Buck creek area, in Clay county, North Carolina, bears a considerable proportion of chlorite derived from the alteration of the amphibole-constituent; and, in places, the olivine entirely disappears, and we have a massive chlorite-rock as a phase of amphibole-picrite. Great masses of chlorite also occur at the Track Rock corundum-mine, in Union county, Georgia, and the origin is doubtless the same.

Besides these rocks, which are intimately connected with peridotite, there occur in some localities, as in Jackson county, North Carolina, numerous chlorite-belts, which seem to have no connection with olivine-rocks. Some of these are massive, though the schistose structure predominates; and their dimensions vary from a few feet to a thousand feet in width. They are traceable across the country, also, for considerable distances, and show no tendency to lenticular form. In two or three places in the Jackson county locality mentioned, the chlorite bears corundum in considerable quantity, and has been the object of some prospecting. These rocks are generally

known locally by the name "blue soapstone," and, indeed, sometimes seem to bear a considerable proportion of talc. Aside from the presence of corundum, I have found no evidence of the possible connection of these chlorite-rocks with peridotite.

Corundum.

While the value of corundum as an abrasive depends on its superior hardness, yet this value is greatly modified, or even sometimes almost destroyed, by certain variations in crystallographic structure; hence, it is deemed advisable to review briefly the more important points of its character, in order to understand fully the problems encountered by the miner.

Character and Varieties.—Corundum, the hardest of all natural substances except the diamond, crystallizes in the rhombohedral division of the hexagonal system. The six-sided prism, however, is frequently well developed, and the crystal seems to have the full hexagonal symmetry, especially when terminated by pyramidal forms. But the rhombohedron sometimes truncates the alternate corners of the prism, and this form is frequently seen where the rhombohedral parting is developed. The forms are best seen on the smaller crystals, which are more perfectly developed, while the large ones are rough, and often corrugated. But the commercial fitness of corundum is most affected by the development of partings. These are usually called "cleavage," but the mineral is not known to possess a natural cleavage, the division-planes being mechanically produced, or due to multiple twinning. Notwithstanding this fact, most corundum crystals do show more or less parting, but the effect is very different from that produced by cleavage. The difference may be expressed, though not explained, in a very few words. The theory will be found in any physical mineralogy, and need not be discussed here.

Cleavage is an inherent property of a crystallized mineral, by which it is capable of being divided into any number of thin sections parallel to certain planes. These planes are those of least cohesion between the physical molecules, and, theoretically, the separation may be effected through any point within the crystalline individual. In other words, cleavage is infinite capacity for being split up parallel to a given plane or planes.

Parting, on the other hand, is finite in its capacity. However

highly developed the parting, the mineral may be split into slips which cannot be divided again along a parallel plane. Parting is a system of parallel planes, developed by mechanical means, or by the natural process of multiple twinning, and the planes have definite positions in the crystal at finite distances apart. The portions between two such contiguous planes have no such capacity for division, and represent the condition of a crystal in which no parting has been developed.

From what has been said, I am sure the distinction is apparent, as well as its bearing on the adaptability of the crushed material for abrasive purposes. Where but little parting is developed, the intervening slips are thick, and large solid grains are obtainable. When greatly developed, the parting becomes, for practical purposes, a cleavage, and much loss is sustained in the process of crushing by the excessive production of very fine sizes and "flour."

Two forms of parting are common in corundum, namely, the rhombohedral and the basal. That parallel to the rhombohedron is more commonly seen, and the effect is a tendency to break the crystal into a number of nearly cubical blocks; the rhombohedral planes forming angles of 93° 56′. The other form, parallel to the base, is often seen in the tendency of crystals to break across into a number of button-like segments. It is also found sometimes in the lamination of irregular crystalline masses, which part along only one plane. Corundum without parting breaks with a rough uneven fracture, and is quite tough.

Miners recognize three forms of corundum, known respectively as "crystal-," "sand-" and "block"-corundum. That which comes from the mine in the form of hexagonal prisms about ½-inch or more in diameter is designated "*crystal-corundum;*" the small crystals or fragments of crystals and irregular grains are called "*sand-corundum,*" and that in large irregular masses, or composed of an aggregation of crystals and grains, and often bearing considerable impurity, is the "*block-corundum.*"

The most variable property of corundum is its color; gray, green, rose, ruby-red, emerald-green, sapphire-blue, dark blue, violet, brown, yellow, and of intervening shades and colorless, is the list ascribed by Mr. George F. Kunz to North Carolina

corundum.* Gray and tints of blue and red are most commonly seen, especially blue, which is often mottled and banded with white or gray and sometimes with red. It is usually somewhat translucent, but transparent corundum is quite rare in the Appalachian field.

The specific gravity is 3.9 to 4.1, and this is one of its most useful properties from the miner's point of view, being utilized in all processes for cleaning and concentration. Of the great number of minerals associated with it, only two common ones, chromite and magnetite, are heavier; garnets and spinels have about the same specific gravity; olivine, chlorite, hornblende, tourmaline and margarite are not so heavy, while quartz, feldspar, serpentine and talc are considerably lighter.

Professor Dana gives "three subdivisions of the species prominently recognized in the arts, and until early in this century regarded as distinct species, but which actually differ only in purity and state of crystallization or structure."† These varieties are sapphire, corundum and emery. Sapphire includes all varieties of sufficient color and transparency for gems. Corundum, as the term is used in the arts, includes the dark or dull, non-transparent varieties, usually light blue, gray, brown and black. Emery is the intimate mixture of granular corundum and magnetite or hematite.

The classes here called sapphire and corundum have been and are still the objects of considerable interest and search in portions of the field under consideration; but emery has been found only at three localities, one each in Massachusetts, New York and North Carolina. These will be referred to in somewhat more detail later.

The alteration of corundum offers an extensive and interesting field for investigation, as may be seen from the work of Dr. Genth along this line. It is a field which I have not attempted to enter beyond such observations as I have made at the mines. I have not seen enough to establish all of the conclusions arrived at by Dr. Genth, but a very little observation in some localities will, I think, convince any one that the alteration of corundum is by no means a rare phenomenon. The frequent development along the parting-planes and over the surface, as

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* *Mineral Resources of the United States*, 1892, p. 760.

† Dana's *System of Mineralogy*, 1892, p. 212.

a radiating sheath, of muscovite and margarite, presents some of the clearest cases. Further work on this subject has just been undertaken for the North Carolina Geological Survey by Mr. Joseph H. Pratt, of Yale University.

The Gem-Varieties.—Except a little search in Sussex county, New Jersey, I have found no record of mining for corundum gems outside of North Carolina. Though not a gem-producing State, there has been considerable interest in this field since the earliest discoveries, and a number of very fair gems have been found. These varieties were the chief attraction of the early explorers and miners, and Corundum Hill—the story of which constitutes the greater part of the history of corundum-mining—was opened and worked for a number of years chiefly as a gem-mine. The importance of the product, and that of other localities in the same region, is indicated in the reports of Mr. George F. Kunz.

"Many specimens," says Mr. Kunz, "have been cut and mounted, especially of the blue and red shades, and make good gems, though not of the first quality. Several rubies of 1 carat each have been found; a blue sapphire 1 carat in weight is in the United States National Museum, at Washington, and a series of fine red and blue crystals has been deposited there by Dr. H. S. Lucas."* Of the peculiar brown crystals with chatoyant lustre, Mr. Kunz says: "These (when cut *en cabochon*) all show a slight bronze play of light, and under artificial light they show well-defined stars, being really asterias or star-sapphires, and not cat's-eyes, as might seem at first sight to be the case." † In 1893 Mr. Kunz expresses the opinion that "the finding of small rubies of fairly good color in Macon county, North Carolina, gives ground for the belief that larger and better stones may be found there by more extended development."‡ Thus, while the output of corundum-gems has not attained commercial importance, it is interesting to note that real gems are being found, and that considerable activity is now manifested in the search for paying deposits of this class.

Commercial Corundum.—This variety is now the only product of the mines, and therefore the one that chiefly concerns us

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* *Mineral Resources of the United States*, 1892, pp. 760, 761.

† *Gems and Precious Stones of North America*, p. 47.

‡ *Mineral Resources of the United States*, 1893, p. 680.

at present. The mineralogical description given above applies in the main to the commercial variety, and as the remainder of this paper will be devoted to its consideration, no further description here is deemed necessary.

Emery.—This variety has been found in Guilford county, N. C., in Westchester county, N. Y., and at Chester, Mass.; but it is doubtful whether any one of these localities properly belongs to the peridotite belt. The North Carolina locality surely does not; the New York emery is developed in an olivine-bearing rock, which presents peridotite phases, but the character of the rocks at this place is very different from that in the well determined members of the belt further south. The Chester, Mass., emery is associated with chlorite and talc-schists and magnetic iron-ore, and may possibly be related to the peridotite of the Appalachian belt, but no traces of olivine-rocks have yet been found there.

Modes of Occurrence of Corundum.

As a rock-constituent, corundum is much more widely distributed than we are accustomed to think. Besides being the chief constituent of emery, Professor Zirkel enumerates the following rocks in which it is found as an accessory; granites, gneisses, granular limestones, dolomites, amphibolites of north-western Austrian Silesia, chlorite-schist in the Urals, graphite of Lower Austria, and as blue sapphire in several basalts, where it is perhaps derived from molten inclusions. He also notes that it occurs as a contact-product of the diorites of Klausen in Tyrol, either as inclusion or accessory in certain eruptive rocks, with cordierite, andalusite, and spinel; as inclusions in andesite and tonalite; in the contact-product of quartz-mica-diorite on quartz-phyllite in Val Moja, and in the kersantite of Michaelstein, Harz.*

Of all these occurrences, emery alone has attained any importance from an economic standpoint. But it is interesting to note that several other modes of occurrence mentioned in this list (as in gneisses, granular limestones, amphibolites, and chlorite-schist), are found in the region traversed by the peridotite-corundum belt under consideration; and that the most impor-

* F. Zirkel, *Lehrbuch der Petrographie,* 1893, page 416.

tant deposits of this region are those in which the corundum does not occur as a rock-constituent, and are therefore not included in Professor Zirkel's enumeration. In fact, while the deposits with peridotite are by far the most important yet developed, still there has been considerable prospecting and some actual mining in others of an entirely different nature. The various modes of occurrence observed in this region will be separately considered in the following order; in chlorite-schist; in amphibolite; in gneiss; in crystalline limestone; in gravel deposits; and associated with peridotites.

Besides the associations indicated in this list, corundum is found in cyanite in a number of places, as in Transylvania and Buncombe counties, N. C., and at Litchfield, Conn. But this mode of occurrence is of only mineralogic importance, and hence has not received a place in the classification. The same might also be said of the corundum in the crystalline limestone, as it really is not a source of any commercial product; but at least some attempt has been made to exploit it for gems, and hence it finds a place in the list, along with others which as yet are not actively mined; no mining being done, at the present writing, except in the peridotite.

Corundum in Chlorite-Schist.—In this case the corundum seems to be a true constituent of the rock. The occurrence has been observed in but few localities. That on the Caney fork of Tuckaseegee river, in Jackson county, N. C., and a similar occurrence near Powder Springs, in Cobb county, Ga., are perhaps the most important. The development of this rock has been referred to in the descriptions of rocks of the peridotite belt. Through the compact fine-scaly chlorite, in the Jackson county locality, corundum is scattered in rounded nodules, ranging in size from mere grains to about one inch in diameter, and always encased in a sheath of radiating muscovite scales, arranged perpendicular to the surface of the corundum. This coating is very variable, being sometimes quite thin and sometimes entirely replacing the corundum, or containing only the minutest fragment. Its secondary nature and its development at the expense of the corundum can scarcely be questioned. The nodules thickly stud one of these chlorite-belts, having a width of about 8 feet. None of the other places have been prospected in such a manner that the deposits, if such exist,

may be seen. This corundum always has the rhombohedral parting.

The emery of Chester, Mass., is found in a chlorite-schist, bordered on the east by talc-schist, the emery-vein sometimes approaching to the border between these rocks. The chlorite has an average width of about 20 feet, is sometimes quite talcose, sometimes filled with clusters of tourmaline crystals, and bears the emery in the form of lenticular masses, standing nearly vertical, and separated from the chlorite on all sides by slickensides. Diaspore and margarite occur, especially at the edges of the lenses, and considerable magnetite without corundum is sometimes encountered.

Corundum in Amphibolite.—The occurrence of pink and ruby-colored corundum in the beautiful grass-green amphibolite of Clay county, N. C., and Towns county, Ga., has been mentioned in the description of that rock, and also the great variability in the dimensions of this constituent—from microscopic grains to broad plates of 3 or 4 inches. The minute grains, as seen by the microscope, are sometimes quite colorless, but they are invariably intergrown with picotite, and inclose very minute specks of that mineral. The larger grains and masses present no such phenomenon to the naked eye. On account of the great difficulty of mining, crushing and cleaning such an intensely tough rock, this is not to be considered as a commercial source of corundum.

Dr. H. S. Lucas has recently brought to light considerable masses of hornblende-rock on Skenah and Cartoogachaye creeks, in Macon county, N. C., in which corundum in small grains and laminated masses is sometimes thickly disseminated. The unaltered rock is chiefly a dark green hornblende in interlocking crystals and blades, but near the surface this is largely altered into chlorite. Besides corundum, magnetite is often abundant, and this has caused the name *emery* to be applied to the formation, though such a usage is manifestly incorrect. In the hydrated surface-material the corundum grains are incased in a zone of white mineral, probably margarite, and often this material entirely replaces the corundum in such a manner as to leave little doubt of its origin from the alteration of that mineral. Tests are being made in processes of cleaning this corundum, and Dr. Lucas has erected a mill at Franklin for its manufacture.

Corundum in hornblende-rock has recently been discovered on the western slopes of Chunky Gal mountain, in Clay county, N. C. The corundum is in grains and small cleavable masses of white, gray, yellow and pink colors, and is thickly dissemimated through portions of the rock. The fragments are usually surrounded by thin coatings of margarite, but sometimes this mineral is developed to the total displacement of the corundum. The rock has been prospected over a width of 150 feet and to a depth of 20 feet, and is quite easily worked with pick and shovel, the hornblende being decomposed to a soft, clayey mass, though still retaining its original structure. Rounded "boulders of decomposition" of forellenstein are occasionally found through the hornblende-rock, and they probably bear much the same relation to it that similar rocks at Buck creek do to the peridotite. A cleaning-plant is now being erected for the operation of this mine on a commercial scale.

In Iredell county, N. C., corundum occurs in zones of vermiculite through the joints and about the borders of a coarse, dark green amphibolite. These zones vary from a few inches to 3 or 4 feet in thickness, and are no less variable in the proportion of corundum they contain, though this bears no ratio to the width of the zones. They consist of fine, scaly, brown vermiculite, bearing the corundum in crystals and lumps with more or less margarite; and sometimes they inclose a vein of feldspar or pegmatite, which is also laden with corundum. In the work thus far done, however, the greater portion of corundum is found in the vermiculite. Between these corundum-bearing zones and the amphibolites there is developed a sheath of actinolite, the parallel fibers and needles of which stand perpendicular to the surface of the rock. On breaking through this casing of actinolite the enclosed rock is often found to have been entirely decomposed, leaving only a mass of ocherous clay, through which occasional needles of actinolite and scales of vermiculite are scattered.

The hornblende-rock of the Presley mine, in Haywood county, N. C., is similar to that of Iredell county, just described, though most of it is schistose and friable. But the corundum is found only in veins of pegmatite, which cut through it, with no development of chlorite or vermiculite zones. The corundum is

surrounded by mica or feldspar and in some cases by margarite. In many specimens these minerals seem to have been produced by alteration of corundum.

Corundum in Gneiss.—In the Appalachian crystalline belt, corundum occurs in a number of localities in the ordinary gneisses and mica-schists of the region. Some of these places are near the peridotite belt, but none of them have any apparent connection with those rocks. In some of the localities the occurrence of nodules of corundum is the only noticeable peculiarity of the rock, its appearance in every other respect being the same as that of the surrounding region in which this mineral is not found. In Jackson and Clay counties, N.C., the corundum, in all the specimens I have seen, is wrapped in a sheath of mica similar to that in the chlorite-rocks described above, and the variation in size of the nodules is about the same. The corundum also has sometimes entirely disappeared from these nodules, only a cluster of mica remaining. The weathered surfaces of the rocks are often thickly studded with the nodules standing out in strong relief on a background of the less resistant minerals.

Dikes of fine-grained hypersthene-hornblende rock cut the gneisses on either side of the corundum in one of the Clay county localities, and the corundum is also found in a zone of fine scaly vermiculite beside one of these dikes.

Along the borders of Cleveland and Burke counties, N. C., in the South mountains, the rocks are highly garnetiferous, gneissic mica-schists, and corundum in grayish blue, tapering crystals is found in pockets of 100 to 200 pounds. The crystals are encased in a tough, white mineral, forming a radiating sheath about ¼-inch thick, which Dr. Genth has called fibrolite.* Crystals of a similar nature are found in the gneiss-regions of Laurens county, S. C., and in Mitchell, Haywood and Macon counties, N. C. Mr. Francis P. King, of the Georgia Geological Survey, found corundum in the gneisses and schists of Forsyth county.†

In 1890 Dr. Genth described the occurrence of corundum in the gneissic garnetiferous mica-schists of Patrick county,

* *Bulletin* 74, U. S. Geol. Survey, 1891, p. 30.

† *Bulletin* 2, Geol. Survey of Georgia, 1894, page 101.

Va. It is associated with andalusite, cyanite, chloritoid muscovite, etc. The corundum is found in crystals on the surface near granite dikes.*

Except the Walker mine in Clay county, N. C., none of these gneiss-localities have been worked. In 1893 about 6 tons of cleaned corundum were taken out there, but most of it was carried away by high water. Although some cleaning-appliances were constructed there, the work was more of the nature of prospecting than mining. A very little desultory work has been done at other places, but none of them have yet given promise of profitable returns.

Corundum in Crystalline Limestone.—This mode of occurrence is found at Newton and Vernon, in Sussex county, N. J., and in Orange county, N. Y., where corundum is found in the limestone along its contact with the gneiss. Mr. Kunz says that several pieces from Vernon have been cut, but it is doubtful if any were true gems.† Specimens of corundum are also found in the white limestone of Orange county, N. Y. As yet, no commercial importance attaches to this mode of occurrence.

Corundum in Gravel-Deposits.—Many of the corundum-deposits with peridotite have been found through the discovery of fragments in the streams and gravel-beds. But such discoveries have not always led to the discovery of the source. Stream-deposits of this kind, in Macon county, N. C., carrying ruby corundum—some of which cuts to real gems—are now vigorously worked, and the watershed is being searched with a view to finding their origin. It is well known that gravel-deposits are still the source of the oriental corundum-gems, though in some cases the original matrix has been found. The gravels represent the concentrates of ages of washing, the stream-beds acting as natural sluice-systems to retain the heavier and more resistant minerals. Hence, while these beds may be profitably worked, the original source may be far too poor in gems ever to become a commercial source. The abundance of corundum crystals in the soil of some gneiss-regions may be due to the same cause, and this supposition will also account for the difficulty sometimes experienced in finding the mineral in place.

Corundum with Peridotite.—Reference has already been made

* *Am. Jour. Sci.*, 3, xxxix., 1890, pages 47, 48.

† *Mineral Resources of the U. S.*, 1882, page 485.

in a number of places to this mode of occurrence as the most common and by far the most important from an economic point of view. Under this head will fall the chief deposits of Pennsylvania, North Carolina and Georgia. In some of these, corundum is found with serpentine, in others with pyroxenite, and, in a still greater number of localities, with peridotite; but they are all essentially the same, the corundum occurring typically in the border-zones of chlorite or vermiculite, or of these two minerals together. The other associated minerals differ considerably in different localities, imparting local character to the deposits; but in all cases the association of corundum and the chloritic minerals is the essential and invariable feature. From the outer borders these chlorite-zones pass into the joints within the peridotite mass, and form a great network of aluminous minerals throughout the non-aluminous olivine-rock.

On account of the great variations in their character from place to place, it is impossible to give a typical section across one of these so-called veins, but the Corundum Hill type will serve to illustrate their nature. In the outer zones we have the following succession: (1) peridotite, more or less altered; (2) enstatite casing, with the fibers and slender crystals somewhat interlocking, but arranged in a general way perpendicular to the peridotite surface; (3) chlorite and vermiculite in varying proportions, usually in coarse plates and wholly irregular in their arrangement; (4) a band of the same carrying corundum; (5) a thin sheet of fine scaly brown vermiculite, with a schistose arrangement against the gneiss or mica-schist. The enstatite casing completely incloses the great jointed blocks of peridotite; and when these blocks are small and decomposed, it may be shelled off in the form of huge pots. Sometimes the peridotite is completely decomposed, only a soft mass of ocher filling the space; but in some cases there remains a kernel of solid rock in the middle of this mass. Smaller nodules are found in which there is only a remnant (or none at all) of the original olivine-rock, showing that the enstatite has grown at the expense of the peridotite. The thickness of this casing at Corundum Hill is usually about 6 or 8 inches; but in different localities it is found to vary from less than 1 inch to 18 or 20 inches.

The chlorite zone is usually barren on either side, bearing

corundum only in the middle or interior portions; in thickness it is far more variable than the enstatite, sometimes almost entirely pinching out, and then swelling to the dimensions of 8 or 10 feet; but these extremes are both, doubtless, partly the result of shearing. The unctuous minerals of the zones would hardly offer any resistance to such action, and evidences of it are often seen in slickensides, especially on the talc-enstatite casing of the peridotite. Much of the parting-development in corundum may also be attributable to this cause. When these zones pass into the peridotite, they become symmetrical by the repetition of the enstatite casing on the opposite sides of the chlorite, but in other respects they are identical with those of the borders, though generally not so thick toward the interior portions.

The corundum is present in small grains or lumps, and in crystals of great variation in size, up to hundreds of pounds, though rarely more than an inch or two in diameter. They penetrate the chlorite to a length of a foot or more sometimes; but their basal parting renders it impossible to remove them whole, and they usually fall into short segments, almost at a touch. The proportions of corundum vary exceedingly, and entirely without regard to the amount of chlorite; the zone may be thickly studded with corundum in its broadest portion, or the corundum-bearing part may dwindle to a narrow strip in the middle and disappear, only to be encountered again after much barren material has been handled. In other words, the occurrence is extremely "pockety;" and whatever other characteristics the deposits may assume locally, this uncertainty always remains. To this character, at least in part, is to be attributed the abandonment of many places that have been prospected or even mined to some extent. Deposits which encouraged the beginning of operations would, perhaps, soon give out; and the work would end in discouragement. But, while some good deposits may have been abandoned prematurely, there has doubtless been far more loss from hasty investment on the "showing" of a rich pocket found near the surface.

Besides the chief constituents enumerated above, there is always present in the corundum-bearing zones a variety of other minerals in smaller amounts. Actinolite is quite common, and tourmaline, spinel, and magnetite are also frequently seen.

Quartz and feldspar are abundant in some localities, and a great many rarer minerals (diaspore, staurolite, anthophyllite, etc.), have been found, but never in any considerable quantity. Corundum is often found also in the gneiss wall of these zones to a distance of several feet; and such material has been worked for more than a year in one of the drifts at Corundum Hill.

One of the most common variations from the type of "corundum-vein" just described, is that found at the Laurel Creek mine in Rabun county, Ga., also at Buck creek and Shooting creek, in Clay county, N. C., and in a number of the Pennsylvania localities. It consists in the presence, in the midst of the chlorite, of a vein of feldspar, which often bears a large percentage of corundum. The Pennsylvania feldspar seems to be albite, while that of Buck creek has the specific gravity and optical properties of anorthite. The corundum in these feldspar-veins is rarely in crystalline form, but usually in irregular lumps, and sometimes perfectly massive in appearance. Usually, however, a high degree of parting is developed, and the corundum bears a close resemblance to the feldspar itself. In this association zoisite, margarite, and black hornblende are often highly developed. Sometimes, though rarely with the peridotite, quartz and mica enter, to form with the feldspar a true pegmatite.

At the Carter mine, in Madison county, N. C., the pink corundum is intergrown in irregular masses with feldspar and massive greenish-black spinel, in a zone of vermiculite and chlorite developed in the midst of the peridotite. Masses of spinel octahedra with corundum are also found at Corundum Hill.

Distribution of Peridotite and Corundum.

There are numerous serpentine-deposits throughout New England and New York, but it would be difficult to determine whether or not they should be considered as the northeastward projection of the Appalachian peridotites. The serpentines of Hoboken, Staten Island, eastern Massachusetts, and Deer Island, Maine, lie in the direct line of this belt; but the continuity is more broken than in the southern Appalachians, and serious structural complications enter which might divert the belt further west, or broaden it to include the whole region. Detailed

studies on these serpentines may bring to light points of similarity by which such a connection could be determined; but corundum is not known to occur with any of them, and hence, for our present purpose, they may be left out of consideration.

The occurrence of corundum with peridotite at Pelham, Mass., seems to come nearer the southern type, and is the only known locality of the kind north of North Carolina. The emery localities at Chester, Mass., and in Westchester county, N. Y., have been already described, and need only be named here. The corundum with cyanite at Litchfield, Conn., and in granular limestone in Orange county, N. Y., and Sussex county, N. J., have also been referred to before. Of the occurrence north of Pennsylvania, only the emery-deposits have assumed economic importance.

South of New Jersey, the peridotite-corundum belt is well defined through Pennsylvania, Maryland, Virginia, North and South Carolina, Georgia, and into central Alabama. Both the peridotite and corundum attain their greatest development in the southwestern portion of North Carolina, near the Georgia boundary. Here the belt is represented by numerous outcrops over a width of nearly 40 miles, and here also are found the largest peridotite masses in the belt; the one at Buck creek being half a square mile in area, and that at Webster nearly twice as large, though considerably drawn out. It must be borne in mind, however, that the quantity of corundum bears no relation to that of the peridotite. The whole outcrop at Corundum Hill, which has furnished a steady output of 200 to 300 tons of corundum a year for nearly twenty years, covers an area of less than one-fiftieth of a square mile (about 10 acres).

Pennsylvania.—In this State the serpentines form an almost unbroken belt from near the Delaware river at Trenton, through the counties of Bucks, Montgomery, Delaware, Chester and Lancaster, to the Maryland border.

In Delaware county, near Village Green, a bronze corundum is found wrapped in, and sometimes replaced by, margarite. Near Black Horse corundum is found at numerous localities over the serpentine area in crystals and masses, loose in the soil. It is also found as crystals in feldspar, some of which are bronze-colored and asteriated on the basal plane. The serpentine mass known as Mineral Hill also furnishes crystals of cor-

undum in feldspar, and large masses associated with margarite and fibrolite.

Dr. Genth mentions a mass of laminated white and blue corundum from the serpentine near the chrome-mines at Texas, in Lancaster county. The only place that has really been mined, however, is near Unionville, in Newlin township, Chester county. Many large masses have been found over the surface here—one, according to Mr. Jefferis, weighing over 5000 pounds. Several tons were taken from shallow cuts and pits, and in one of these was found a large lenticular mass 7 by 14 feet and 54 feet in depth. This consisted chiefly of corundum and margarite, with associated feldspar, spinel, diaspore, etc.*

Besides serpentine, pyroxenites (sometimes olivine-bearing) are found along this belt in Pennsylvania, and some of the serpentines are thought to have been derived from these rocks. A peculiar type occurs between Chestnut Hill and Bryn Mawr, for which Dr. H. Carvill Lewis suggested the name "*perido-steatite.*" It consists of talc, in which are imbedded numerous nodules of serpentine, often bearing a striking resemblance in form to the crossed twins of staurolite. In fact, they have been considered as pseudomorphs after this mineral; but the microscope shows that they still contain a considerable proportion of unaltered olivine. Hence this type is practically identical with the olivine-bearing talc of the Urals, called *glinkite.* We shall note below a similar rock in Ashe, Alleghany and Madison counties, N. C. It may be added that there are reasons for believing that this is only an altered form of harzburgite—the olivine-enstatite rock—but discussion of the evidence cannot be entered into here.

Mr. Theo. D. Rand, of Philadelphia, informs me that several tons of corundum in crystals and masses, some quite large, have been obtained from a decomposed gneissic rock in Lehigh county, Pa.

Maryland.—The serpentine-belts, as represented on the geological map of Maryland, are strikingly regular; but there are numerous breaks, both in this State and in Pennsylvania, where the masses are mapped as continuous. The chromite-deposits,

* "Mineral Localities of Chester County," by W. W. Jefferis, *Proc. Acad. Nat. Sci. Phila.*, 1892, p. 188.

which once assumed great importance along the Maryland-Pennsylvania line, are found at intervals through Maryland; in fact, the location of the chrome-works at Baltimore is due to the deposits with serpentine at the State line, Bare Hills, and Soldiers' Delight, which were intended to supply the ore. None of these places are now worked.

No deposits of corundum are known in Maryland. Tyson mentions it as occurring near White Hall; but the report has not been verified by later investigation. The magnesian belt passes about 20 miles northwest of Washington and crosses the Potomac River above Great Falls.

Virginia.—Rogers reports a number of serpentine and talc outcrops across Virginia, the belt passing near Lynchburg and crossing the North Carolina line just west of the Blue Ridge. Only two occurrences of corundum have been reported from this State, namely, a single crystal from Louisa county, 40 miles north of Richmond,* and surface-specimens in the gneiss-region of Patrick county, already described under "Modes of Occurrence."

I have not yet found olivine-rocks reported from Virginia, though this type is represented at the State line, where it passes into North Carolina. Here it is chiefly a talc-olivine rock, similar to the glinkite ("peridosteatite") described above from Pennsylvania.

North Carolina.—In North Carolina, the magnesian rocks through Alleghany and Ashe counties are chiefly soapstone derived from enstatite-rock, and harzburgite in which the enstatite is mostly changed to talc, while the olivine remains fresh. Enstatite-rock is also plentiful in Watauga county; and the fresh pure olivine type, dunite, is found just north of Boone. Near Cranberry there is serpentine and then dunite, harzburgite, and enstatite-rock through Mitchell and Yancey counties. In Mitchell county the first corundum in the peridotite-belt since leaving Pennsylvania is found with enstatite-rock; also an occurrence in gneiss is found near Bakersville. In Yancey county, it occurs again on the South Toe river with enstatite-rock, and in the extreme western part of the county, near the Tennessee line, it is found with peridotite in what is called the Egypt (or Hayes)

* G. F. Kunz, *Mineral Resources of the U. S.*, 1883-84, p. 735.

mine. Here I obtained the only specimens of corundum I have seen *imbedded in the dunite itself.* From this point to Haywood county, the outcrops are scattered over a width of about 15 miles.

From southern Yancey almost across Madison county, the belt is chiefly serpentine, with visible grains of olivine in places, and one fresh outcrop of dunite. The Carter mine is in this county, near the Buncombe line, and is situated at the northern extremity of a laminated dunite outcrop which extends southward into Buncombe for a distance of 3 miles. A crystal weighing 46 pounds was found above this mine on the surface of the gneiss. Here two other parallel lines of outcrop are defined: a small one 2 miles to the eastward, and another of somewhat more importance, just west of Marshall, consisting of talc with occasional harzburgite (or *glinkite*—olivine in talc). Near the mouth of Ivy river, in Madison county, corundum occurs with hornblende-rock. Large crystals are found on the surface—one weighing 17 pounds—but none have yet been found in place. It was in this county, 3 miles below Marshall on the French Broad river, that the first piece of corundum was found in North Carolina, nearly fifty years ago—a surface-specimen of laminated blue corundum, near a talc-chlorite strip which here represents the third parallel line of outcrop above referred to. It is worthy of note also that this locality has furnished but one other specimen.

The principal line of outcrop crosses the French Broad river just above Alexander; the peridotite reverting to serpentine again before reaching the river. With slight exceptions, the serpentine and talc are the only representatives of the belt in Buncombe county.

This brings us to Haywood county, where corundum is found in three distinct associations; namely, with peridotite, on the north fork of Hominy creek; in pegmatite through amphibolite, at the Presley mine; and in gneiss, on the west fork of Pigeon river. The peridotite is confined to the locality mentioned, though talc is found westward as far as the Pigeon river. The next peridotite outcrop is found a mile west of Balsam gap; and from this point to Webster the line is almost continuous, leading to the eastern border of the great elliptical zone lying northeast of Webster. The laminated peridotite, and the gneisses both within and without this figure,

have an outward dip, presenting perfectly the form of an eroded dome. The greater axis of the ellipse lies in a northeast and southwest direction, and is about 6 miles long, while the shorter diameter is 3½ miles. The dimensions of the peridotite-outcrop vary exceedingly, it being about one-fourth of a mile in width at Webster, but thinning to a narrow strip of talc in several places. On the eastern border there are several complete breaks, and the isolated outcrops have the typical lenticular form. Near the middle of the broadest portion, at Webster, is found the type-locality of the pyroxene rock, websterite. No corundum is found here, and scarcely a trace of chlorite, but a little of both occurs at Addie, the northeastern extremity of the ellipse. Peridotites occur also in the southeastern portion of the county, and considerable mining of corundum has been done at Sapphire. The occurrence of corundum in chlorite-rocks in this county has already been described, and needs no further mention here. Some gravel-deposits have also attracted attention through the hope of finding gems.

Some of the outcrops worked at the Sapphire mine are in Transylvania county, and corundum is further found in enstatite-rock on the headwaters of the French Broad river.

In Macon county, the principal locality, as well as the oldest and most successful corundum mine in the belt, is at Corundum Hill, 7 miles east of Franklin, on the Cullasaja river. The peridotite at this place covers an area of about 10 acres, and has an oval form, its longer axis lying in the direction of the strike of the gneisses. Two finger-like branches are given off on the western side, passing into the gneiss in a direction parallel to the strike; otherwise the outline is smooth, and the ends are bluntly rounded. The surrounding rock is gneiss; but on the eastern border mica-schist is developed. The mining and mode of occurrence of corundum here have been already described.

An area of about 30 square miles here, lying to the north of Corundum Hill, is thickly studded with similar outcrops, all of which bear corundum. Much prospecting has been done, and considerable capital invested, with a view to opening new mines, and this work still goes on; but no mining proper has yet been inaugurated. Two places on Ellijay creek were worked for about three years in connection with Corundum Hill, but are now abandoned. The line of peridotite and talc follows up

the Little Tennessee river into Rabun county, Ga., and a little corundum is found in one place on Fish Hawk mountain. Nine miles west of Franklin, corundum crystals are found over the surface of the gneiss on Cartoogachaye creek. The gravel-beds of Cowee creek, northwest of Franklin, carry crystals and grains of gray and ruby corundum, along with pink and wine-colored garnets and a number of other minerals; vigorous work is now being done there in the search for the source of the ruby corundum.

The great Buck creek outcrop is the largest in Clay county, and the largest compact mass of peridotite in the belt, having an area of about 300 acres. There is a larger area in the Webster outcrop, as already mentioned, but there it is drawn out to a considerable length. The Buck creek area is very irregular in outline, somewhat approximating a triangle, and narrow strips pass off into the gneiss on two sides. The main mass is traversed by several dike-like strips of the grass-green amphibolite (the so-called "smaragdite") described above in the classification of the rocks. Toward the north side also, a considerable portion of the peridotite is amphibole-picrite or the chloritic rock derived from it; and at the western extremity a small amount of serpentine has been developed. One of the most interesting phases of the peridotite, from a petrographic point of view, is the forellenstein, which constitutes a portion of an arm projecting into the gneiss along the eastern border. The only phases that bear corundum, however, are the amphibolite (in which it is a constituent of the rock) and the dunite, which constitutes the greater portion of the outcrop, and in which the corundum is found in chlorite-zones and veins of anorthite and coarse black hornblende.

On the head-waters of Shooting creek, in Clay county, corundum occurs with peridotite along a line of outcrop that passes near Bell Knob in Towns county, Ga. It is also found in the gneiss near hypersthenite dikes and pegmatite on Thumping creek, and in amphibolite on the western slope of Chunky Gal mountain. Further down, near the mouth of the creek, is the Behr mine, where the corundum is associated with peridotite; and in the immediate vicinity to the southward it is found with considerable feldspar and zoisite, and in nodules wrapped in margarite. The grass-green amphibolite with red corundum is also found near the Behr mine.

In addition to the corundum localities named above in tracing the peridotite belt, may be mentioned the important amphibolites in Iredell county, those of Skenah and Cartoogachaye creeks, in Macon, that on Chunky Gal mountain in Clay, and the Presley mine in Haywood counties. These have already been sufficiently described under "Modes of Occurrence," and are mentioned here only for the purpose of completing the list of localities. In gneissic rocks corundum is found in Gaston county; near Bakersville in Mitchell county; near Carpenter's Knob, in Cleveland and Burke; on Turkey Knob, on the Macon-Jackson county-line; on Cartoogachaye creek in Macon county; and on the head-waters of Shooting creek in Clay county. Emery is found in the gneiss-region west of Greensboro in Guilford county. Associated with interlocking bladed cyanite, corundum occurs on Owens creek in Transylvania county, and at Swannanoa gap in Buncombe. A number of the localities named have been prospected, and a little mining has been done in the gneisses and amphibolites.

South Carolina.—Peridotite and corundum are found in Oconee county, a few miles east of the Laurel creek mine of Georgia. The outcrops are almost continuous with those in the region of the Sapphire mines and Whitewater river, N. C.; but they have received very little attention from prospectors, and none, as yet, from geologists; hence their extent and distribution are unknown. I have seen corundum-specimens said to have come from Pickens county; and crystals associated with talc fragments are found near Gaffney, in Spartanburg county. Fine encased crystals are found over the surface of the mica-schist area for 3 miles north-east of the court-house, in Laurens county.* Corundum is also reported from Anderson county, but its associations there are not known.

Georgia.—The peridotite-outcrops, so far as determined in Georgia, do not form so complete a belt as in North Carolina, but they are found scattered in a broken, irregular manner through the counties lying in the same direction, and passing southwesterly along the valley of the Chattahoochee river.

From Clay county, N. C., the peridotite passes almost continuously into Towns county, Ga., at two points. The most westerly lies along the upper portion of Brasstown creek, and

* Joseph Willcox, *Proc. Acad. Nat. Sci.*, Philadelphia, 1878, page 159.

carries corundum in several places, and the one further east is found with the corundum-deposits of Bell creek. The former is continued, except for short interruptions, to Track Rock mine, in Union county. Here the corundum is found in a large outcrop consisting chiefly of green hornblende and chlorite, though some fragments of olivine rocks have also been found.*

The zone of peridotite lying in the valley of the Little Tennessee river passes into Rabun county north of Clayton; corundum is known in a few localities, though the principal interest of this region attaches to the asbestos, which, in most cases, seems to be a fibrous enstatite.

The chief corundum-mine of Georgia lies very near the corner of this State with the Carolinas, and is known as the Laurel creek (or Pine mountain) mine. Some idea of its relative importance may be had from the historical sketch in the beginning of this paper. This outcrop is very similar to that at Corundum Hill; the general form is the same, and the rock-types represented are practically so, though the proportion of enstatite-rock is much greater at Laurel creek. The mineralogical features, while similar in some respects, are very unlike in many leading characters. Corundum occurs in the chlorite and vermiculite zones, and the "sand-vein" of this nature was one of the chief sources while the mine was worked. A large proportion of corundum also occurs in feldspar-veins lying within these border-zones of chlorite; and great blocks of this nature were mined, some of it perfectly massive in appearance and exceedingly tough.

The strike of the gneisses of this region, and the general trend of the peridotite-corundum belt, would place the deposits of Laurel Creek and vicinity in the same zone, or sub-belt, as the Sapphire mines in North Carolina.

Corundum is known with hornblende-rock near Porter Springs, Lumpkin county; and it occurs in chloritic rocks, in nodules wrapped in margarite, in Habersham. Corundum, also wrapped in margarite, is found in chlorite- and enstatite-rocks near Gainesville. In White county, peridotite is known, but no corundum has been found. Surface-specimens have

* For most of the information that follows in regard to Georgia localities, I am indebted to *Bulletin* 2, of the Geological Survey of Georgia, "A Preliminary Report on the Corundum Deposits," by Francis P. King. 1894.

been picked up in a region of mica-schist and hornblende-gneiss, in Forsyth county. A corundum, with a high development of parting, and said to be an inferior abrasive, is found in a region of deep decomposition in the vicinity of Acworth, in Paulding county. Surface-specimens are scattered over small areas in several places in Cobb county, which adjoins Paulding on the east. One chlorite-schist vein, near Powder Springs, has been found to carry corundum in abundance, in small grains and crystals.

In Carroll county, corundum is found near Villa Rica on the surface, with an outcrop of actinolite-talc and peridotite; and corundum incased with margarite, similar to that at Gainesville, is found near Carrollton, the county-seat. An occurrence in black hornblende is reported from Heard county; and it is found in small amounts at several asbestos-localities in Troup county. The localities thus far mentioned lie within the Chattahooche river belt, which passes west of Atlanta; but basic magnesian rocks are also found in Walton county, about forty miles due east of Atlanta. From the surface of a talc-actinolite outcrop near Monroe, 500 pounds of "block-corundum" have been gathered and shipped. Some prospecting has been done here for asbestos, but none for corundum.

Alabama.—In Alabama, the crystalline belt passes under fhe Cretaceous and later formations of the coastal plain in the central part of the State, near Montgomery. Peridotites occur in the vicinity of Dudleyville, in Tallapoosa county, and corundum has been found in surface-fragments in this, and in Coosa, the adjoining county to the west. Careful search would doubtless reveal the presence of peridotite, and perhaps also of corundum, to the very borders of the crystalline rocks.

Mining Methods.

Twenty-five years ago, the corundum-fields under consideration were entirely new to the mining world. Corundum itself, as a commercial article, was known only in the forms of emery and the gems; as an associate of peridotite it was not even known to mineralogy. There were no precedents in mining; and every step had to be evolved by the slow and expensive process of experiment. Undue excitement had been aroused by the finding of a few gems, and the idea that corundum might

be profitably mined as an ordinary abrasive had scarcely been conceived. Under these conditions, it is not surprising that the first undertaking (at Corundum Hill) was a failure. A hydraulic method, with short sluice-systems, was adopted for working the corundum-bearing soil and gravels, and some of the chlorite-zones, or "veins," were also opened. Only the larger crystals and promising gem-material were saved. The precious pieces were cut for use in jewelry, or were sold with the finer crystals for cabinet-specimens; the rest was sold chiefly to the manufacturers of dentists' tools. Gems, however, were the chief product, and, as they were too scarce to render the work profitable, the mine was soon abandoned.

In this early mining, and in most of the early prospecting, two great mistakes were made—first, too much importance was attached to gem-mining; and, secondly, the smaller fragments, or "sand-corundum," which afterwards became the leading product, were entirely overlooked. These points were pretty clearly demonstrated in this early work at Corundum Hill, and in small attempts in other localities during the first few years after the discovery. When Dr. Lucas reopened Corundum Hill, it was solely for the purpose of mining an abrasive, and methods were adopted by which all material would be saved. On this basis, the mine has been successfully operated without interruption to the present time.

Prospecting.

Accidental discoveries of corundum-fragments, as in the case of the first mines, still continue to give the clue to many of the new localities that are brought to light every year. But in the peridotite-deposits, which include by far the greater number of corundum-localities, it was soon found that, amid great and constant differences, there were certain characters and mineral associations common to nearly all; and these characters are successfully used by the prospecter in the search for new deposits.

In the principal corundum-region, in the south-western counties of North Carolina, the great majority of the people are well acquainted with the various forms of corundum, and the more common minerals found in the mines. Some of these men devote a considerable portion of their time to the search for

new localities, or the prospecting of old ones, and thus has arisen a class of semi-professional prospecters, or "mineral-men," as they are called.

Loose fragments of corundum in the soil or stream beds, while the surest indication of corundum-bearing deposits, are not always the most easily traced to their origin. The extreme resistance of this mineral to the ordinary processes of abrasion and decomposition render it, when exposed at the surface, almost indestructible. Hence it may be transported for great distances down mountain slopes and streams without showing any appreciable alteration or wear. When associated with fragments of peridotite, chlorite or talc this floating corundum is a much more valuable guide. In such cases it has probably been transported only a short distance, and the search for the source becomes far less difficult. The rock-fragments indicated are, in themselves, valuable guides to the deposits; but where no corundum occurs with them, and none is found by panning the soil or sands, as in the ordinary search for gold, the prospect for finding valuable deposits is not very encouraging.

The "indications," of whatever nature, are followed up the grade by which they would most naturally have reached their present position. If found in a stream, search is made up the stream and its tributaries till fragments are no longer found, and then up the adjoining hill-slopes till the parent-mass of peridotite is reached. The borders of the outcrop are first examined. If they are not readily found by inspection, ditches are cut through the soil at right angles to the strike of the rocks. The corundum will probably be found in the zone of chlorite and vermiculite along the border between the gneiss and peridotite. This border should be exposed by a ditch and explored at intervals by pits or open cuts. When no encouraging deposits are found in this way, ditches may be cut across the peridotite mass, in search for joint-zones of corundum-bearing chlorite; but these are usually less developed than those of the border.

In some places margarite is found abundantly with corundum; but in the great majority of cases this is not true, and its discovery usually follows rather than precedes that of the corundum. The discovery of the emery-mine at Chester, Mas-

sachusetts, however, was due to the association of this mineral; and when found, it is to be regarded as a good "indication."

Obviously, the points above mentioned apply chiefly to deposits associated with peridotite. Corundum is found, however, in a number of other associations in the Appalachian region, and several localities with amphibolite and gneiss give promise of becoming commercially important. To such occurrences there seems to be no reliable index except the finding of corundum itself; and where it is developed in considerable quantity this should not be difficult.

The necessity for thorough and intelligent exploration before building mills or making investments cannot be too much emphasized. To one not acquainted with the actual conditions, such advice seems entirely superfluous; it is simply common-sense, such as any intelligent man should be able to exercise. But a 10-days' tour through the principal corundum-region will sufficiently demonstrate that this advice has often been sadly needed.

Mining.

The methods of work in the few places that have been really mined have varied considerably, according to location, character of the material mined, nature of the walls, etc. But most of the area under consideration is in mountainous or very hilly districts, and the outcrops are usually on high ground, with good natural drainage. These conditions and the general loose texture of the border chlorite-zones—which are the principal deposits worked—have led, in the majority of cases, to the adoption of open cuts and drifts. While the work is still near the surface, the open cut is found to be the most convenient and economic method; but with increasing depth, the peridotite-wall, with the development of unctuous magnesian minerals along the joints, is exceedingly liable to cave in. Trouble also arises from the same cause in the drift-workings, but it may be remedied by careful timbering. Shafts have usually been resorted to only in low or flat places, as in the principal working at Laurel Creek, in Rabun county, Ga.; but the principal work at Unionville, Pa., was conducted in this manner. They are objectionable chiefly on account of the expense incurred in hoisting and pumping.

In most cases the material handled is scaly chlorite and vermiculite, with disseminated corundum, and is easily removed with a pick and shovel. In solid feldspar-veins, however—as in some of the workings at the Unionville, Sapphire, Laurel Creek and Buck Creek mines—blasting has to be employed, and the material must be afterward crushed for cleaning. In one of the drifts at Corundum Hill the work is also dependent on blasting, as the material in this case is the gneiss-wall adjoining the peridotite, and is very hard and thickly impregnated with corundum. Hand-cars and wheel-barrows are employed in removing the material from the mines. It is then, according to its nature, dumped into wagons or into a current of water in troughs, to be carried to the mill for cleaning. At Corundum Hill the loose material is thus carried in troughs 1½ miles.

Cleaning.

The differences in cleaning-methods are even greater than those of mining, and are constantly varied according to the character of the material. The cleaning of the "sand-corundum" may be illustrated by describing the method in use at Corundum Hill. The material, as already described, consists of loose chlorite and vermiculite plates and scales, bearing small crystals and irregular masses of corundum, varying in size from fine sand to occasional pieces of an inch or more in diameter. Troughs about a foot square carry a small stream of water, and are sufficiently inclined to give a swift current throughout the distance traversed. Into these is thrown the loose material from the mine. Over the course of a mile and a half this material is subjected to vigorous scouring and rolling by the current, and in several places is given a vertical drop of 8 or 10 feet.

Arriving at the mill, the coarser material is separated and crushed with rolls till it will pass through a sieve of 14 meshes to the inch. It is then placed in sluices with high cleats and a strong current of water and stirred with hoes till most of the lighter material is floated off. From these it is transferred to chasers or mullers, which consist of shallow tubs with two heavy wooden rollers propelled around the circumference by arms carried on an upright shaft. A current of water is introduced at the bottom and constantly carries off the finely-divided light

materials over the top, while the corundum remains. In this machine, the corundum-grains clean each other by their own cutting power on the softer minerals adhering to them. After three to five hours of this treatment, according to the nature of the impurities, the corundum is removed and dried by dropping about 20 feet down a smokestack and then sliding down an inclined plane through the flame in the furnace. It is then screened of all material coarser than No. 14 and sacked for shipment to Chester, Mass., where it is crushed and sorted into sizes for market. The finer numbers are subjected to a final cleaning process by passing through a magnetic separator for the removal of the magnetite.

The tough hornblende- and feldspar-veins encountered in some of the sapphire-workings, and the hard corundum-bearing gneiss at Corundum Hill obviously cannot be treated in this manner. These materials, when taken out in large blocks, are broken by the very primitive method of building a fire over them till they are heated, and then suddenly cooling them by throwing on water. The fragments are then passed through rock-crushers and rolls till they are brought to the size desired for cleaning. At Corundum Hill, this is fourteen to the inch, as with the "sand-corundum." The crushed material is now wetted and passed through a machine with a spiral worm, like a great auger, by which it is rolled over and over and scoured upon itself for some time. It is then stirred in the sluice-boxes with hoes and placed in the mullers for final cleaning, as described above.

Various designs have been used for cleaning-machinery, and some of the prospecting is now done with appliances different from those described here. But the same principles are applied in all; namely, gravity, and the scouring action of the corundum particles upon each other.

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MISCELLANEOUS MENTION.

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Gold-Milling in the Black Hills, South Dakota, and at Grass Valley, California.

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(Atlanta Meeting, October, 1895.)

OUR *Transactions* contain two notable papers descriptive of the stamp-milling practice of the Black Hills and of Grass Valley, namely, the elaborate and complete treatise of Prof. H. O. Hofman, on "Gold-Milling in the Black Hills,"* and the paper describing the North Star mill, by E. R. Abadie, its superintendent.† Having visited and examined the stamp-mills of both localities, I venture to offer here such comment as more recent information or a different point of view has suggested.‡

* *Trans.*, xvii., 498.

† *Id.*, xxiv., 208.

‡ For more detailed descriptions and discussions, I would refer to a series of

I.—The Black Hills.

Prof. Hofman's paper was prepared in 1888, seven years ago, yet the methods in vogue to-day at Lead City and Terraville do not differ materially from those so carefully described by him.

The mining industry of the Black Hills is not so actively prosperous in 1895 as it was in 1888. Not so many stamps are dropping; some of the mines have been compelled to suspend operations owing to a falling off in the yield of ore, while others have passed into the comprehensive control of the Homestake management. Nevertheless, the mines and mills of the Belt produce more gold than any other single mining camp in the United States, with the exception of Cripple Creek, Colorado.

The Belt is that part of the region contiguous to the Homestake lode and its extensions, reaching from Whitewood creek to Deadwood gulch. It is the center of the mining activity of the Black Hills, the only important mining region in the State of South Dakota.

The geological relations of the ores of the district have been referred to in our *Transactions* by W. B. Devereux* and F. R. Carpenter, respectively.† It is to be regretted that so important and interesting an ore-deposit as the Homestake vein should not have long ago undergone more detailed description at the hands of some one of our members. The commercial environment of mining enterprises often militates against scientific investigation.

While the writer has not had an opportunity of making a careful examination of the mines, a visit underground was sufficient to show the justification for the great ore-reduction establishments of the Homestake Mining Co. The ore occurs in large bodies of quartzified chloritic schists, conforming to the structure of the country and being a portion of it. The width of milling-ore varies from 50 to 400 feet. Most of the

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papers on "Variations in the Milling of Gold-Ores," which I have contributed to the *Engineering and Mining Journal* during the present year, and from which quotations are made, and statistical material is reproduced in the present paper. The illustrations herewith given will be found, with others, in the *Journal* of September 7 and 14, 1895, and are here used through the courtesy of the Scientific Publishing Company.

* *Trans.*, x., 465.

† *Id.*, xvii., 570.

supply comes at present from above the 500-foot level; but the developments in lower levels, down to the 800-foot, indicate the continuity of the enormous ore-shoots of the mine. The huge excavations made along the outcrop of the lode (See Fig. 1), are a distinctive feature of the present topography of the Belt, and bear impressive witness to the immense quantities of mill-stuff that have gone under the stamps. Extensive bodies of ore are still held in reserve in these surface open-cuts.

Behind the Highland shaft-house there is a large cut, which gives a very excellent section of the geological formation, showing the uptilted edges of the gold-bearing schists, overlain by Cambrian sandstone. The latter, only a few feet thick at this point, is split by an intruding sheet of the porphyry which also overlies the whole formation and caps the hilltop. The lowermost member of the Cambrian series is a conglomerate, identified by its fossils as belonging to the Potsdam period, and said* to have been derived from the degradation of the gold-bearing lode under the outcrop of which it was formed by the seas of a very early geological time. These facts indicate for the Homestake vein an origin of remarkable geological antiquity.

The porphyry above mentioned is a felsite. At one time it was broken with the ore and sent to the mills in spite of its valueless character. Now it is used for filling up stopes. Underground it can be seen in dikes conformable to the vein-walls, and splitting the ore-bodies by its intrusion.

The first mill in the Black Hills was that erected by the Racine company, at the lower end of Lead City, in April, 1877. The beginning of the mining industry of the Black Hills dates back to June, 1876, when the Wheeler brothers found rich gravel in Deadwood gulch. The outcrops of the large quartz-lodes were early seen, but quickly discarded by the California and Montana miners, who ridiculed the idea of the profitable handling of ores which yielded on panning only from three to ten dollars per ton.

After the placer-mines had commenced to yield handsomely, and while the quartz-lodes were unappreciated, the early min-

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* By Devereux, *Trans.*, x., 466, *et seq.* The development of the conglomerates of the Witwatersrand adds much interest to these Dakota deposits.

ing activity was diverted to the development of the lowermost beds of the Potsdam formation, the gold of which lay in a conglomerate. It was to reduce this conglomerate that the first stamp-mills were introduced. They were of the primitive Colorado pattern, and proved entirely unsuited to the extraction of gold from such material. They made a poor record, and were followed by the fast-dropping stamps, modeled on Californian practice, first introduced by Mr. Augustus J. Bowie, Jr., in his design of the Father de Smet mill,* the erection of which, at Terraville, was commenced in June, 1878. This plant was succeeded by the first of the Homestake Co.'s mills, which also had a narrow mortar and a rapid discharge. The Caledonia, erected the following year, used more roomy mortars and two inside amalgamating-places, more after the Colorado fashion. This mill is now idle, chiefly for lack of suitable ore-supply, and therefore a comparison between the two styles of working is not possible. Nevertheless, I do not hesitate to say that the Homestake mortar, deep and narrow, gives a combination for rapid pulverization and high percentage of extraction inside the battery which render it, for the ores of this district, far superior to any other mortar I know.

All the mills now at work on the Belt are under the direction of the Homestake management, with the exception of the little 10-stamp Columbus mill at Central City. When Prof. Hofman wrote his paper, there were 660 stamps dropping on the free-milling ores of the Belt; to-day the number is 550. The De Smet and Caledonia mines have been unable to survive a diminution in the tenor of the ores they produced, and the mills belonging to them have ceased operations, the former in 1892 and the latter in 1893. The Highland Co.'s mill has been lately increased by the addition of 20 stamps. The Golden Terra and Deadwood mills were consolidated six years ago, the 80 stamps of the Terra being placed behind the 80 stamps of the old Deadwood mill. The two large mills of the Homestake Co. have undergone steady enlargement, and in addition to the number of stamps given in the annexed table, there are 40 about to be added to the Golden Star mill, whose total will then be 200 stamps.

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* See *Trans.*, x., 87.

TABLE I.—*The Stamp Mills of the Belt, South Dakota.*

Name.	Date of Erection.	Location.	Number of Stamps.		Owners.
			1888.	1895.	
Homestake........	1878	Lead City.	80	100	The Homestake Mining Co.
Golden Star......	1879	Lead City.	120	160	
Highland..........	1880	Lead City.	120	140	The Highland Mining Co.
Deadwood.........	1879	Terraville.	80	160	The Deadwood-Terra Mining Co.
Golden Terra.....	1880	Terraville.	80		
Father de Smet..	1878	Central City.	100	100	The F. de S. Mining Co.
Caledonia..........	1879	Terraville.	80	80	The Caledonia Min. Co.
Columbus..........	1894	Central City.	......	10	....

The ore is dumped at the shaft's mouth into the rock-breakers. At the time of Prof. Hofman's investigations, all the Homestake mills were using the Blake, and the Caledonia had just introduced a Gates crusher. Since that time the Gates has replaced the Blake rock-breaker in every mill on the Belt. Furthermore, the rock-breaker is now placed in the shaft house instead of at the mill. This follows the tendencies of modern practice in California, where the crusher at the mine delivers the broken ore to the tramway, which carries it to the mill, or sometimes to a second rock-breaker. The latter arrangement relieves the stamps of the hard work of stone breaking, facilitates pulverization in the mortar and gives uniform conditions more favorable to successful amalgamation.

The transference of the rock-breaker from the mill to the mine is in itself a praiseworthy change. It enlarges the capacity of the ore-bins at the mill, and renders unnecessary the use of separate bins for coarse and fine. In small plants where the breaker, if at the mill, would not be driven by a separate engine, it does away with that irregularity in the working of the mill arising from the unequal consumption of power on the part of a rock-breaker. It renders easy the loading of the cars which bring the ore from the mine, a factor important in the case of aerial rope-ways carrying buckets of small capacity. But more important than these minor advantages is the almost entire cessation of the production of the dust so injurious to all the mechanism of the mill and always such a nuisance to those who work in it.

Most of the mills use the No. 6 Gates breaker. The Dead-

wood-Terra and Highland have two each. The Homestake mine has three, one of which is held in reserve to avoid delay due to repairs on either of the others. Experience has shown that the larger receiving capacity of the Gates and its greater crushing power render it more suitable for large milling-establishments than the Blake.

Prof. Hofman has described the various methods for transmitting the power to the different parts of the mill. Of these, the arrangement in the Golden Star mill resembles that in vogue in California and in the best Colorado mills. The driving-shaft is approximately level with the cam-shaft, and the connecting-belts are nearly horizontal. The latter are in a place easily accessible, well lighted and away from the dirt and water inseparable from the close neighborhood of the battery itself. Such an arrangement requires that the sills under the cam-floor shall be made stronger than if they simply supported the flooring, but the additional expense is trifling compared to the convenience of the plan. On the other hand, the placing of the counter-shaft immediately underneath the feeder-floor, in addition to the inaccessibility and inconvenience, the environment of dirt and water, the absence of light, etc., requires the use of tighteners injurious to the belting. The Star mill uses one belt while the Highland wears out three.

All the batteries are placed upon flat sites in two rows back to back, save at the Father de Smet, where the two rows of batteries face each other and discharge toward the center of the mill. The latter arrangement gives a larger storage-capacity to the bins overhead, but this advantage is obtained at the greater cost of darkening the amalgamating-tables.

The mortar is the most interesting feature of the Homestake mills. In Prof. Hofman's paper there are drawings of it, one of which is reproduced here in Fig. 2. Fig. 3, from a photograph furnished by Fraser and Chalmers, illustrates the latest design. The changes in the dimensions made since 1888, the date of Prof. Hofman's paper, are as follows:

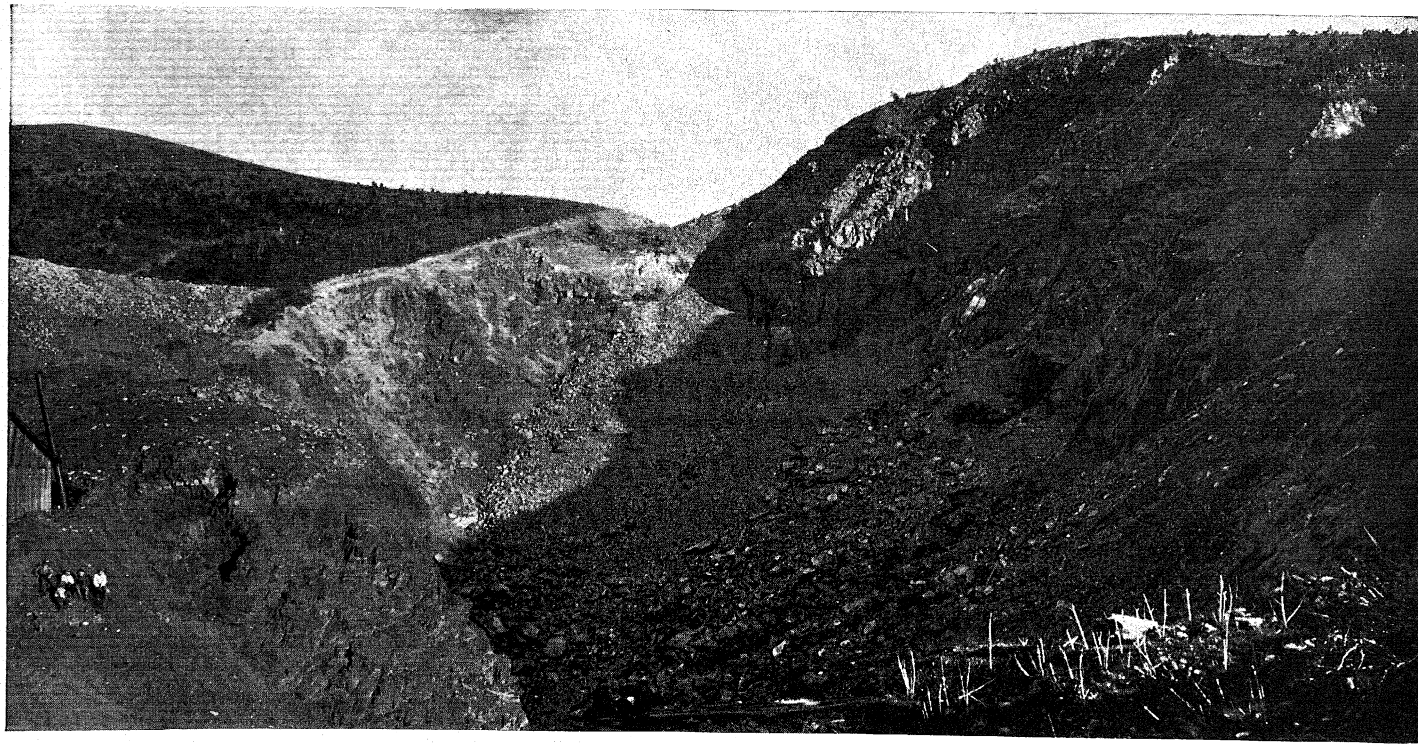
	1888.	1895.
Weight, ...	5400 pounds.	7300 pounds.
Length of base, ...	54¾ inches.	56¾ inches.
Width " "	27¼ "	28¼ "
Height, ...	54½ "	58¼ "
Inside width at the level of the lip,	13½ "	12½ "

The most important change is the narrowing of the interior width at the level of the lip of the mortar, where a slight change is more important than in any other dimension. The measurement of a new mortar lying outside the Golden Star mill gave an inside width of 13 inches. At the Columbus mill it is 12 inches. The mortar, as now made, is provided with cast-steel false bottoms $2\frac{1}{2}$ inches thick, with a cast-iron lining $\frac{7}{8}$ inch thick along the sides and $\frac{1}{2}$ inch thick upon the feed-hopper. The inside copper plates, placed along the front, are $\frac{1}{4}$ inch thick and are attached to chuck-blocks. The latter are wooden blocks, designed so as to serve as a false lip to the mortar, thereby raising the depth of the issue. A piece of 2-inch plank is bolted to a $1\frac{3}{4}$-inch board, the latter being made to project about 2 inches beyond the former, to which the copper plate is attached. The 2-inch plank had been replaced, at the time of Prof. Hofman's inspection, with $\frac{1}{4}$-inch iron, but has since been reverted to, because the slight increase in the distance between the chuck-block and the shoe obtained by the arrangement he describes was undesirable in a mortar characteristically narrow and designed for rapid crushing.

Two of these chuck-blocks are in use, one 9 and the other 7 inches high. When new dies have been introduced, the former is inserted, then making the distance from the bottom of the screen to the top of the die about 9 inches. As the dies wear down, the depth of discharge increases until, after about a fortnight, it becomes necessary to replace the high chuck-block with the lower one. The difference of 2 inches between the two is approximately equal to the diminution in the thickness of the die. After a further service of two or three weeks the dies are worn out, the depth of discharge has increased to 11 inches, new dies are inserted, and the high chuck-block is re-introduced. In this way some sort of an effort is made to maintain a rough uniformity in the depth of the issue, a factor the importance of which is generally overlooked or underestimated in stamp-milling.

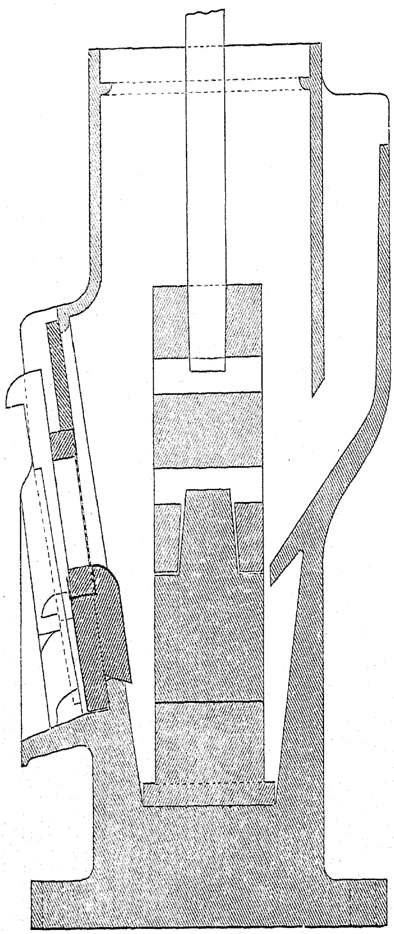
It may be added that the copper plate on the high chuck-block is flat, while on the other it has a curved surface, and is mounted on slightly thicker wood, so as to bring it nearer the die. It is the intention of the mill-man to keep the bottom of the chuck-block about on a level with the bottom of the shoe,

Fig. 1.



Deadwood-Terra Open Cut.

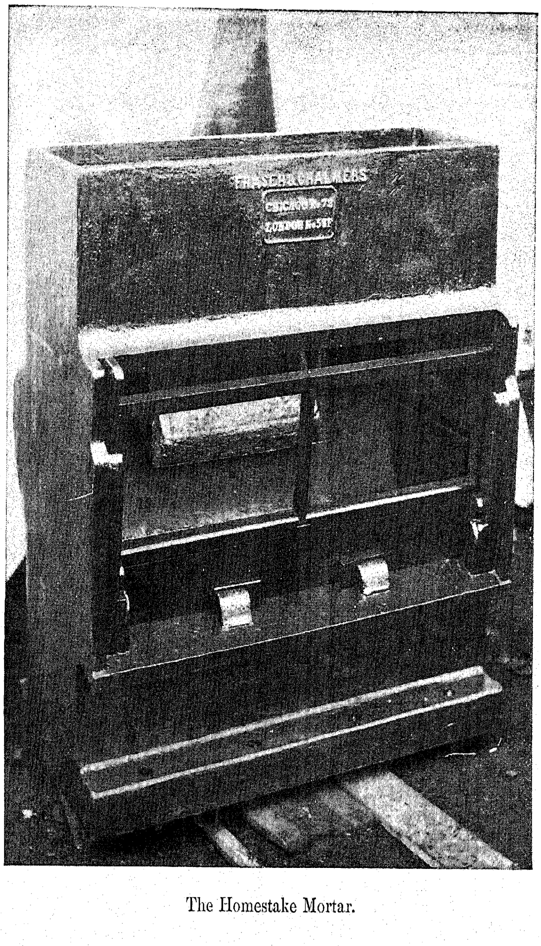
Fig. 2.



Scale, 1 inch = 1 foot.

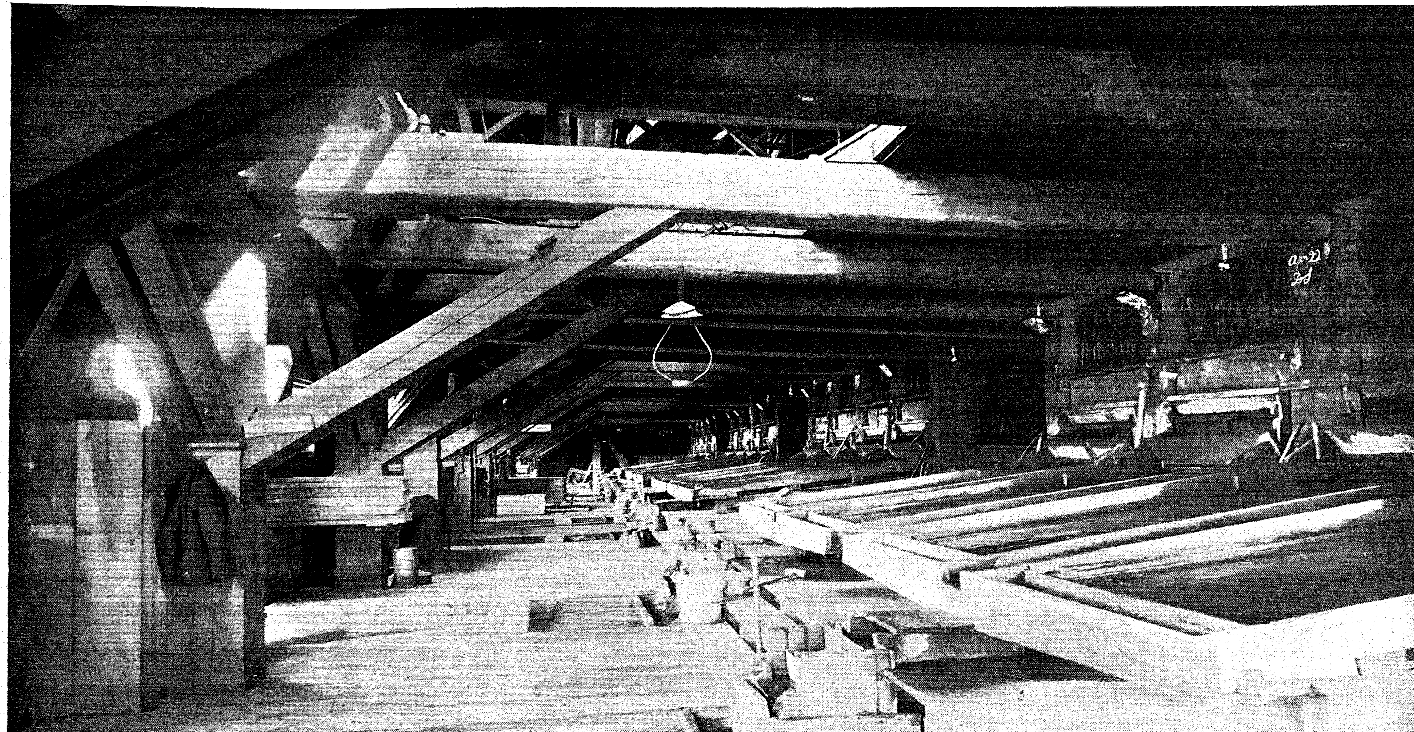
The Homestake Mortar.

FIG. 3.



The Homestake Mortar.

FIG. 4.



Interior View of the Golden Star Mill.

and to avoid so close a neighborhood to the ore on the die as would lead to the scouring of the amalgamated surface of the copper plate.

There is only one inside plate, made of plain copper and 5 inches wide. This is the one attached to the chuck-block. There is no back-plate, as was the case in the Caledonia mill (now idle), where the mortar was more roomy and of a different design. At intervals, free mercury is fed with the ore into the battery.

The Homestake mortar combines, to a notable degree, the two excellent features of a rapid discharge and a high percentage of amalgamation. Its width at the issue used to be 13½, was then diminished to 13, and in the newest design is 12 inches. The depth, by the introduction of chuck-blocks, is raised to from 9 to 11 inches. The mortar becomes thereby both narrower and deeper than the Californian pattern, its narrowness compelling a rapid expulsion of the pulp and giving the mill a capacity nearly twice that of the average Californian battery when working ore of similar character. At the same time the depth of the mortar prevents the scouring of the inside plate, and permits the arrest of the gold by this plate and by the free mercury added with the ore, so that the percentage of extraction follows closely in the wake of the roomy mortar of the Colorado mill, the crushing capacity of which is only one-quarter that of the Homestake. The following comparison will be of interest:

TABLE II.—*Comparison of Typical Mills.*

	Width at Issue.	Depth of Discharge.	Weight of Stamp.	Number of Drops per Minute.	Height of Drop.	Crushing-Capacity per Stamp per 24 hours.
	Inches.	Inches.	Lbs.		Inches.	Tons.
Golden Star, Deadwood, S. Dak......	12½	9 to 11	850	85	9½	4
Hidden Treasure, Black Hawk, Col..	24	13 to 15	550	30	17	1
North Star, Grass Valley, Cal........	17½	4	850	84	7	1½
Pearl, Bendigo, Australia.............	15	3¼	840	74	7¼	2¼

It will be seen how closely the crushing-capacity is related to the interior width of the mortar at the level of the issue.

Notwithstanding its rapid crushing, the Homestake mortar retains a percentage of the total gold extracted which compares well with any of the other districts, and is superior to some of them, though this factor will be affected by the variety of screen in use.

In 1888 the Homestake mills were using No. 7 diagonal-slot Russia iron punched screens. At the present time the mills uniformly employ the No. 8 size of the same variety. This means slightly finer pulverization. Prof. Hofman gives the crushing-capacity in 1888 at $4\frac{1}{2}$ tons per stamp per day, with a 9-inch drop 85 times per minute. It is now about 4 tons per day, with a $9\frac{1}{2}$-inch drop and the same speed. This shows the effect of the substitution of the No. 8 for the No. 7 screen.

The No. 8 screens are considered equivalent to 30-mesh wire. They break before they become worn out because of lines of weakness developed by the press used in their manufacture. They are, however, never retained in service until the apertures are much worn, because this would produce a coarse crushing detrimental to a uniform product. Their maximum service is about two weeks, but this is rarely attained, because breakage occurs after six to eight days. The cause of this is the accumulation, inside the battery, of wood chips, which have found their way into the ore from the timbers in the mine. They tend to dam up the pulp within the mortar, and so subject the screen to a pressure greater than it can bear. This results in a break or a burst. Where surface-ore is being milled, as at the Deadwood-Terra, the life of the screen is prolonged to an average of nine or ten days, because such ore comes from workings where there is but little timbering.

At the neighboring Columbus mill, 30-mesh brass wire is used, and the choking of the screens is minimized by having three sets for each battery, so that while one is in place the second is being dried out and the third cleaned with a wire brush. It would be better if the Homestake mills could find it practicable to use wire-cloth instead of punched Russia iron, since, apart from the advantage of a more uniform discharge, the pulverization is more regular, because the wires do not wear so easily as the punched holes, and the apertures therefore retain their size, and the screen does good work until simple breakage requires that it be patched or discarded.

The process of gold-extraction consists of amalgamation within the mortar upon outside plates and in traps, supplemented in a rudimentary way by an inadequate effort to concentrate the sulphides of the tailings.

The mortar becomes an amalgamating apparatus by the use of the inside copper plate and the addition of free mercury. About 50 per cent. of the total amalgam is obtained from these inside plates. At the Deadwood-Terra the proportion reaches 70 per cent.

To quote from my previous paper* on the subject:

"Mercury is fed into the battery in quantities proportioned to the richness of the ore and regulated by the condition of the amalgam on the apron-plates. At the four principal mills the rate at which the mercury is fed can be judged by the accompanying record, covering the two weeks previous to my visit.

"It will be observed that though the Golden Star and the Deadwood-Terra mills crush approximately the same amount of ore, the former uses more than twice as much mercury. This fact is explained by the wide difference in the richness of the ore, for while that crushed in the batteries of the Golden Star averages from $4 to $5 per ton, that which goes through the Deadwood-Terra ranges from $1.50 to $2 per ton.

TABLE III.—*Consumption of Mercury at the Homestake Stamp-Mills.*

Date.	Deadwood-Terra. 160 Stamps.		Golden Star. 160 Stamps.		Highland. 120 Stamps.		Homestake. 100 Stamps.	
	Lbs.	Ozs.	Lbs.	Ozs.	Lbs.	Ozs.	Lbs.	Ozs.
May 1, 1895	11	5	25	13	15	4	† 9	10
2,	10	2	† 17	14	† 12	9	12	2
3,	10	0	25	16	15	1	13	0
4,	12	2	26	13	17	10	12	12
5,	† 9	12	25	3	10	6	11	7
6,	10	6	24	8	16	8	12	2
7,	11	11	23	2	17	0	12	8
8,	11	6	24	12	19	3	12	15
9,	11	8	23	13	19	8	12	13
10,	10	14	25	3	18	12	12	13
11,	11	0	23	7	18	8	12	7
12,	10	12	24	6	17	4	11	1
Average per day	10	14	24	3	16	5	12	2

"For the year ending June 1, 1894, the Homestake mill used 2084 pounds and the Golden Star mill 3440 pounds, making a total of 5524 pounds avoirdupois,

* *Eng. and Mining Journal*, p. 222, September 7, 1895.

† Indicates clean-up days.

which, at the price obtaining that year, 42 cents, makes the value of the mercury used $2320.08. During that time 309,210 tons of ore were crushed, so that the consumption was at the rate of about 5 dwts. Troy or ¾ cent per ton. At the Deadwood-Terra 205 pounds were used in February, 1895, in treating 18,483 tons of ore. It is estimated that 22 per cent. of the amount of mercury used is lost."

On issuing from the battery the pulp falls from 6 to 10 inches before striking the aprons or first amalgamating-tables. This serves no particular purpose, while the actual damage possible to the plates by the scouring of their surface due to the impact of the pulp is obviated by the interposition of a splash-board, which breaks the fall of the sand and water. This splash-board might be placed at such an angle as would permit of its use as an amalgamating device by lining it with a copper plate.

In the Homestake Co.'s mills the aprons are 10 feet in length and 4½ feet wide. Those in the Highland mill are only 8 feet long. In all these mills two apron-plates deliver the pulp to one tail-plate having a size equal to one apron. At the Deadwood-Terra the aprons are somewhat larger, namely, 11 feet by 4 feet 8 inches, but the tail-plate is 8 feet long and only 16 inches wide. The latter is called, very appropriately, a sluice-plate, and is a truly absurd device for arresting the gold.

In the Homestake mills, both apron- and tail-plate have a slope of 1½ inches per foot, the minimum gradient at which the tables can clear themselves of the pulp. Both tail-plate and apron are dressed each morning, the aprons are cleaned up partially each day, and more completely deprived of their amalgam at the bi-monthly general clean-up, when both the tail-plates and the inside mortar plates are gone over.

The traps are intended to arrest escaping amalgam. The Golden Star mill (see Fig. 4) has two at the head of the tail-plate. They are 18 inches deep. They are preceded by a shallow trap or riffle 2 inches deep, which is stated to do better work because of the more regular passage of the pulp. These traps catch about 1 per cent. of the total amalgam. They are, to quote again,

"Cleaned up every two weeks, the accumulated pyrites are shovelled into buckets and then passed into a pan, which extracts all the free amalgam. The residues from the pan are then fed into a particular 5-stamp battery, provided with a No. 10 slot screen. They are passed through this battery twice and are then sent to the smelter, their final assay value being about $38 per ton.

"The above suggests the Australian practice of mercury wells, particularly employed at Clunes, with the obvious difference that the Homestake traps are not supplied with free mercury. It is claimed that if sufficient mercury is fed into the battery no free gold should escape, and the mercury in the traps would merely serve to thin the amalgam and make it easier of escape. The traps catch concentrates and amalgam only."

Then comes the concentration of the sulphides. Of this, not much can be said. The Deadwood-Terra mill makes no attempt to save the sulphides, since the ore comes from near the surface and contains nothing worthy of supplementary treatment. The Homestake mills and that of the Highland company are using the Gilpin county bumping-table. Seven years ago Prof. Hofman, noting the absence of any effort to save the valuable sulphides, suggested the employment of *Spitzlutten*, supplemented by *Spitzkasten*, preparatory to the recrushing of the coarse sands and the concentration, on buddles, of the fine. This very sensible advice has not been followed. Instead, however, two blanket houses were erected, and, without any sizing or classification such as should precede all concentration, the blanketings were worth from $20 to $30 per ton. These have been idle for several years, and in their place eight bumping-tables were placed in both the Highland and Golden Star mills, while six were added to the Homestake mill. It is only necessary to add that the two larger mills have crushing capacities of 560 and 640 tons per day, and the Homestake about 400 tons, to indicate the absurdly disproportionate nature of the equipment, which can only be considered a badly-designed experiment. The results obtained are not by any means a proof of what could be done under proper conditions, as already stated in my previous criticism of this feature:

"During the year ending June 1, 1894, the two Homestake mills produced 915,010 pounds of concentrates whose assay value varied from $5 to $8 per ton. They consist of iron pyrite, arsenical pyrite and pyrrhotite. The ore contains from 3 to 5 per cent of sulphides, but only about 2 per cent. are saved. They are sent by rail to the Deadwood and Delaware Smelter, just below the town of Deadwood, where they are treated at a charge of half their assay value, and converted into an iron matte very low in copper and rich in gold, which goes to the Omaha and Grant Smelting and Refining Company, at Omaha, for further treatment."

At the present time experiments are being made with jigs in order to improve this part of the milling. It is to be regretted

that a representative company, such as that operating the Homestake mines, should be so slow to adopt the best metallurgical practice, remaining satisfied with a manifestly inadequate equipment and a thoroughly unscientific treatment until the successful work of a Cornishman, treating its mill tailings a few miles down the creek, emphasized the desirability of doing something better.

Prof. Hofman has described in detail the method of sampling the mill-stuff by simple panning. At the present time the mill-work is checked by sampling the pulp as it leaves the apron-plates and fills a dipper at intervals of an hour for a period of five hours. This is done each afternoon. The gold is determined by fire-assay. No accurate knowledge of the completeness of the extraction can be obtained by so unsystematic a procedure.

The labor-costs are given in Table IV.

By comparing this with Table V. in Prof. Hofman's paper, it will be noticed that the mills have been enlarged without a proportionate increase of workmen. In 1888 the Homestake mill employed 20½ men to attend to 80 stamps and the Golden Star 23½ men for its 120 stamps. The chief change is in crusher-men, the number of whom, by the substitution of the large Gates for the small Blake rock-breakers, has been diminished from 5 and 6 men respectively to 2 for each mill. The Golden Star now employs 4 general laborers in place of 2, and the Homestake 2 in place of 1, and this is the only increase following the enlargement of the mill. The engine-men, firemen, amalgamators, etc., remain the same in number, while the general superintendence (foreman) has been diminished for any single mill by giving one man charge of both the Homestake mills as well as that of the Highland Co.'s.

The milling costs, per ton of ore, are given in Table IV.

Prof. Hofman gave the costs for the year ending June, 1888.

It is remarkable how little the increased crushing-capacity of the mills, due to additional stamps, has diminished the costs per ton, and this in spite of the general cheapening of material inseparable from the much improved communication between Deadwood and the centers of industry. This applies especially to the Homestake mill. In the case of the larger plant the diminution of 12 cents per ton, as compared with 1887–88, did

TABLE IV.—*Labor Employed in the Principal Stamp-Mills.*

Name of mill / Number of stamps	DEADWOOD-TERRA. 160.			HOMESTAKE. 100.			GOLDEN STAR. 160.			HIGHLAND. 120.		
	Number.	Shift hours.	Wages, dollars.	Number.	Shift hours.	Wages. dollars.	Number.	Shift hours.	Wages, dollars.	Number.	Shift hours.	Wages, dollars.
Foreman	½	......	7.00	⅓	......	8.00	⅓	......	8.00	⅓	......	8.00
Millwright	1	12	3.50	1	12	4.25	1	12	4.25	1	12	4.25
Pipefitter	......	......	......	½	......	3.50	½	......	3.50	......	......	......
Enginemen	2	12	3.00	2	12	3.50	2	12	3.50	2	12	3.50
Firemen	2	12	2.50	2	12	3.00	2	12E	3.00	2	12	3.00
Night foreman	......	......	......	⅓	......	5.00	⅓	......	5.00	⅓	......	5.00
Head amalgamators	2	10	3.50	1	10	4.00	1	10	4.00	1	10	4.00
Amalgamators	4	12	3.00	4	12	3.50	4	12	3.50	4	12	3.50
Crushermen	3	10	2.50	2	10	3.00	2	10	3.00	2	10	3.00
Oilers	2	12	2.50	2	12	3.00	2	12	3.00	2	12	3.00
Feeders	3	12	2.50	2	12	3.00	4	12	3.00	4	12	3.00
Laborers	1	10	2.00	2	10	2.50	4	10	2.50	3	10	2.50
Total	20½	......	59.00	19 1/16	......	64.33	23⅙	......	75.33	21⅔	......	71.08

not follow the addition of 40 stamps, since previous to that the cost had been 3 cents less. Much of this discrepancy is traceable to the fact that the wood, fuel, water, castings, foundry-work, etc., are supplied to the mill by subsidiary companies.

TABLE V.—*Cost of Stamp-Milling in the Black Hills, South Dakota.*

A.—THE HOMESTAKE MILL.			
Year Number of stamps	1887–1888. 80.	1888–1889. 80.	1893–1894. 80.
Tons treated	96,790	106,780	104,995
Labor	$0.2561	$0.2395	$0.2543
Supplies	0.0130	0.0045	0.0105
Water	0.1729	0.1562	0.1986
Wood	0.2766	0.2230	0.0597
Coal	....	....	0.1784
Machinery	0.0922	0.0893	0.1097
Oil	0.0109	0.0084	0.0034
Candles	0.0016	0.0014	....
Quicksilver	0.0103	0.0053	0.0083
Lumber	0.0070	0.0139	0.0167
Timber	....	....	0.0155
Total cost per ton of ore	$0.8406	$0.7415	$0.8551

B.—THE GOLDEN STAR MILL.			
Year Number of stamps	1887–1888. 120.	1888–1889. 120.	1893–1894. 160.
Tons treated	146,565	161,755	204,215
Labor	$0.2138	$0.1755	$0.1556
Supplies	0.0079	0.0044	0.0121
Water	0.1712	0.1622	0.2043
Wood	0.2739	0.1959	0.0346
Coal	....	....	0.1637
Machinery	0.1220	0.1088	0.1057
Oil	0.0084	0.0066	0.0021
Candles	0.0014	0.0014	....
Quicksilver	0.0252	0.0082	0.0071
Lumber	0.0054	0.0088	0.0160
Total cost per ton of ore	$0.8292	$0.6718	$0.7012

As to the efficacy and completeness of the mills as ore-reduction plants, I find myself (always excepting the feeble attempt at concentration) to be very favorably impressed. It has been urged by unfriendly critics* that "haste and waste" is the chief

* Recently by Mr. C. G. Warnford Lock in "Gold Mining and Milling in the

characteristic of the milling-methods; but this has been said, I venture to believe, without proper regard to the conditions of the case.

Something has already been said, in the early part of this communication, regarding the mode of occurrence of the ore. The immense size of the ore-bodies and the extensive nature of the mine developments justify the scale of the Homestake Co.'s operations. The ore is mixed with a large proportion of country-rock—in fact, it is for the most part only gold-bearing schist, without any marked boundary or any very noticeable difference between what is worth exploitation (and consequently ore) and what is unprofitable to mine (and therefore waste or "country"). Of the sulphides, the most favorable association is that of arsenical pyrites. But the amount of sulphides present is small, and the gold is not very closely attached to them, so that the ore is notably "free-milling." In fact, in my opinion, it is more docile than even the quartz of the main Californian gold-belt. The results obtained in the mills confirm this view.

However, it is hard. The wear and tear of shoes and dies indicates this. The shoes wear at the rate of from 36 to 37 pounds per 100 tons of ore, and the dies at the rate of 44 to 48 pounds. This represents a cost of only about 2 cents per ton, at the prices which the Homestake Company pays, as compared to 4 cents at Angels Camp* (California), 5½ cents in Gilpin county (Colorado), 4½ cents at Bendigo and 5½ at Clunes (in Australia), 9 cents at Grass Valley† (California), and 13 cents at Mammoth,‡ Pinal county, Arizona. These figures are instructive, but they depend largely on prices and freights. The following statement of wear in pounds of iron per 100 tons of ore is a better guide: At Angels Camp, 45 pounds; Grass Valley, 90; Gilpin county, 58; Bendigo, 144; Clunes, 157; Mammoth, 76; as against an average of 80½ pounds at the Homestake. The ore of Grass Valley is very hard indeed, while the lesser hardness of the material treated by the Australian mills is much more than offset by the absence of rock-breakers. The ore of Angels Camp and Gilpin county is comparatively soft.

At present the Homestake mills save, as far as I could learn,

Black Hills," read before the Institution of Mining and Metallurgy, London, January 16, 1895.

* In 1891. † In 1892. ‡ In 1893.

somewhere about 75 per cent. The ore is low-grade. During the year ending June, 1894, the Homestake and Golden Star mills treated 309,210 tons, yielding $1,390,610, equivalent to $4.49 per ton. Having in regard the low tenor of the ore, the immense reserves of it and the evident intention not to work for the good of posterity, it seems, from a commercial standpoint, a very proper thing to treat it with the utmost dispatch and rush it through the mills. Of course, slower treatment would give a higher extraction, but this would not compensate for the greater cost per ton due to diminished capacity. Moreover, the results are not bad; 75 per cent. is an extraction above the average even in mills treating a fraction of the quantity crushed per stamp in this district. The after-treatment, the saving of the sulphides, is a serious error, and the arrangement of the plates* is a blemish; but the excellence of the design of the mortar, the ample rock-breaker capacity and the general arrangement of the mills is such that, taken as a whole, the milling practice is one of the best examples of the cheap treatment of a large mass of low-grade ore.

II.—Grass Valley.

Mr. Abadie, in his excellent description of the work at the North Star mill, did not concern himself with the methods of his neighbors at Grass Valley. This was doubtless largely due to the fact that the mill of which he had charge was generally acknowledged as representative of the practice of the district.

The following additional matter, based on visits to this district made in December, 1886, May, 1891, and July, 1893, may, however, render the description of the milling-practice more complete.

The first mill erected in California was not built at Grass Valley in 1850, as stated† by Mr. Abadie. Mariposa county claims that distinction, and accords it to a mill of 8 stamps, each in its own separate mortar, erected on the Mariposa estate late in the summer of 1850.‡ It was not till the following Jan-

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* Placing one tail-plate below two aprons having a combined amalgamating-surface twice as great as the tail-plate.

† *Trans.*, xxiv., 208.

‡ Information derived from Mr. Melville Attwood and others of the pioneers.

uary that a mill was erected on the west bank of Wolf Creek, nearly opposite the site of the present Empire mill at Grass Valley.

Mr. Abadie's drawings and photographs very completely illustrate the position and interior arrangement of the North Star mill. The drawings are particularly valuable. The most notable change made in the North Star mill since my first visit in 1886 has been the new arrangement of the amalgamating-tables. When going through the mill for the first time, I noticed the inadequacy of the amalgamating-surface and the uselessness of the narrow sluice-plates. Since then the shaking-tables have been replaced, in the case of the last two batteries added to the original 30 stamps, by new wide plates which are nearly 16 feet long. The old short apron and narrow sluice-plate have been thrown out. The other six batteries were, at the time of my last visit, and, it would appear,* up to the time of Mr. Abadie's description of the mill, still provided with the primitive apparatus which was first criticized by me nine years ago.

This is the great blemish of the North Star plant, which has followed in this respect the design of the typical Californian mill, whose narrow sluice-plates are a remnant of the apparatus originally borrowed by the quartz-miner from the placer-digger.

In order to discuss this question, a detailed description† of the passage of the pulp on its discharge from the mortar will be necessary. The screen-frame, which is 4 feet 4 inches long and 18 inches wide, has four partitions dividing the issue into five portions. Each division of the screen surface is 9 inches wide and 12¾ inches high. While this construction strengthens the screen, it robs it of 1 square foot of discharging area, and is not therefore to be commended.

The pulp then drops six inches and strikes the battery-plate, which is 4 feet 2 inches wide and 18 inches deep. It covers an iron apron which is bolted to the mortar.

Then there succeeds a trough from which the pulp passes through a distributor, consisting of a vertical iron partition

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* *Trans.*, xxiv., 215.

† Much of which appeared in my article on "Grass Valley," published in the issues of May 19 and 26, and June 2, 1894, of the *Engineering and Mining Journal*, New York.

pierced by 20 $\frac{3}{4}$-inch holes. Then follows a drop of $3\frac{1}{2}$ inches to the apron-plate. The latter is 4 feet 5 inches wide* for $2\frac{1}{2}$ feet, and then becomes narrowed for the remaining 2 feet, finishing with a width of 22 inches. Then come the sluice-plates, 22 inches wide and 12 feet long. They had a slope of 1 inch per foot at the time of my visit, but according to Mr. Abadie, the gradient is usually greater, viz., $1\frac{1}{4}$ inches. The aprons are given $1\frac{1}{2}$ inches per foot.

Then come the copper-lined shaking-tables. The new plates are 16 feet long, of which the upper portion, of $2\frac{1}{2}$ feet, is 53 inches wide and represents the former apron, and the remainder is 46 inches wide, representing an enlargement of the sluices.

Mr. Abadie says that these new plates "cannot be too highly recommended," which is quite true when we contrast them with the old arrangement; but one may be permitted to ask, Why this narrowing of the plate from 53 to 46 inches? I take the liberty of emphasizing this matter because it has for many years seemed to me that the California stamp-mill, otherwise the best machine of its kind yet evolved by the ingenuity of man, suffers seriously from an unsuitable arrangement of the amalgamating surface. In tracing the evolution of the apparatus of the stamp-mill it will be found that the first gold-saving methods were modeled after placer-mining practice, and that in the term "sluice-plate" lingers the evidence of the transference of the sluice-box from the gulch into the mill. The arrangement of 4 feet of plate as wide as the mortar, followed by 10 or 12 feet of plate somewhat under 1 foot wide, was almost universal in California a few years ago, one mill copying another apparently without inquiring into the object of the arrangement. At the North Star an examination of the sluices showed the edges of the copper plate to be abraded or scoured by the swift passage of the pulp.

The philosophy of the sluice-plate is not evident. The battery and apron have caught the coarser gold—that which it is easiest to arrest—and the object is to prevent the escape of particles which, because they were fine and difficult to catch,

* Some of my figures, obtained by actual measurement in the mill, vary slightly from those since given by Mr. Abadie.

have not been stopped. Hence, the sluice-plate is put in, but it is more a launder for the convenient conveyance of the pulp to the vanners than a gold-saving device. How can we expect to catch gold which could not be arrested on a wide plate, by passing it over a very narrow one? The quantity of water and crushed ore is still the same, but it is crowded into a much lessened space, the speed of its passage is increased, and the depth of its flow augmented. In some mills the grade of the sluice-plate is actually greater than that of the apron.

Therefore the ordinary practice should be reversed, the apron should be succeeded by a wider rather than a narrower plate, additional facilities for the catching of the gold should be given by spreading the pulp so that every opportunity is afforded for its contact with the amalgamated surface of the plate.

The necessity for a uniform depth of battery discharge is hardly appreciated at Grass Valley. At the North Star no serious attempt was made to regulate it, so that it used to vary from 2 to 6 inches. Since my last visit it has become the practice to introduce cast-iron plates 2 inches thick underneath the dies. They have lessened the difference between the minimum and maximum depths of discharge, according to Mr. Abadie,* to 2 inches. Although his account does not state the fact, yet the context would indicate that these plates are introduced after the dies have been worn down, thereby restoring the height of the issue. At the Empire old dies used to be placed underneath as the dies in use were worn down, and this was found preferable to employing iron plates for false bottoms (as at the North Star), because the latter broke so often. If they are made to fit snugly, this breakage should be no great detriment, as the pieces will remain in place. In addition to these methods, the Empire mill uses the device of fixing wooden cleats to the bottom of the screen outside. The sand banks up inside the mortar, and protects the unused strip of screen until such time as, the dies having worn down, the cleats are removed and the issue lowered. At another mill, the W. Y. O. D. ("Work Your Own Diggings"), the dies are discarded before

* *Trans.*, vol. xxiv., p. 212.

they have worn down deeply. The remnant is sold to the local foundry, and helps to pay for new dies. This is wise. The consumption of a few pounds of iron is a very small matter compared to the importance of maintaining the conditions best adapted to good work. When the depth of discharge varies between wide limits, the operation of the mill must be irregular. The minimum and the maximum depths cannot be equally favorable to the particular conditions required, and an effort should be made to find the exact depth best adapted to the treatment of the ores of the special mine, and that figure should then be maintained as far as is practicable.

In the matter of screens, the Grass Valley mills have, as it seems to me, taken a retrograde step. The general adoption of punched tin-plate in place of wire-cloth is defended on the ground of economy, Thus, Mr. Abadie says that "the life of a tin screen is about 30 days; the cost, one-fourth that of wire screens." As a rule, at Grass Valley, the brass-wire screens cost $1.55 apiece, while the tin-plate costs 55 cents per screen, the former giving a service of 25 days, the latter of 14 to 15 days. Steel-wire cloth was discarded because that which was used in this district had the defect of a shifting of the horizontal wires. The introduction of tin-plate into the Californian mills dates several years back. I first saw it in use at the Utica mill, Angels Camp, in 1886. In 1893 the Idaho was the only mill in Grass Valley which was not employing tin-plate in preference to wire-cloth. It is the usual custom to burn off the tin upon the blacksmith's forge, with the idea that the iron plate becomes annealed and toughened. Since tin amalgamates, its removal prevents the adherence of mercury to the screens.

The cost of screens per ton of ore varies from $\frac{3}{4}$ to 1 cent per ton, an item of expense which is trifling when compared to the importance of getting a screen which will properly size the pulp and aid in maintaining the conditions most favorable to good amalgamation. As between punched tin-plate and wire-cloth, the advantage in cheapness possessed by the former need not be considered unless it accompanies other more serious advantages; the difference of half a cent this way or that is as nothing when measured against good milling.

No one at Grass Valley had, as far as I could learn, made any

careful experiments to determine the action upon the pulp of the use of the two varieties of screen. One would naturally expect* a more free issue and a more uniform crushing when using wire-cloth in place of punched plate,† because the former has a discharge-area approximately one-half of its surface,‡ while the latter, having apertures of the same size, has less of them per square inch.§

In making such tests it is necessary to be particularly careful, not only that the same kind of ore is fed and that the mortars are the same, but also that the shoes and dies are in the same state of wear, so that the depth of discharge is equal in both batteries. Moreover, attention should be paid to the possibility of more fine stuff finding its way into one mortar than into the other. The outer batteries of a stamp-mill, where the rock-breaker is in the center, receive more than their share of the fines.

The milling-practice of the Grass Valley district has undergone important changes during the past ten years. The introduction of an inside amalgamating-plate—first done at the North Star in 1888—marked an important departure from the extreme type of fast-crushing California battery. The tendency to increase the percentage of inside amalgamation by deepening the discharge and inserting a chuck-block (which also carries an amalgamating-plate) is a notable feature. Thus from a rapid pulverizer the California mill has been made an improved amalgamator. This older, very shallow-drop mill, unable to use a plate inside the mortar, and relying solely on the outside tables, has become rare on the Pacific slope. It has become a recognized fact that rapid crushing will not compensate for poor amalgamation; that the sooner we catch our gold the

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* The only results of experiment published are those contributed by Mr. Thos. H. Leggett to the *Eng. and Mining Journal* of June 30, 1894, where it is shown that as between a round punched tin, No. 0 screen and a 40-mesh steel cloth, the former made nearly 11 per cent. less fines (passing through a 100-mesh screen) than the latter.

† A recent test at the Mammoth mill, Pinal county, Arizona, showed that as between a No. 6 slot-screen and an equivalent wire cloth, 24-mesh and 26 B. W. G. wire, the latter crushed 20 per cent. more than the former.

‡ With a width of mesh .027, a thickness of wire .01, and a gauge (B. W. G.) of 33, the discharge-area is just one-half the total surface.

§ See also *Trans.*, xxiii., 563.

better; and that the best feature of the stamp-mill, as compared to other pulverizers, is its capability to combine the crushing and amalgamating apparatus in one machine.

At the North Star and W. Y. O. D. mills nearly twice as much amalgam is obtained from the inside as from the outside of the mortar; while at the Empire the inside extraction is from 50 to 85 per cent. of the total saving.

Heavy silver-plating is recommended now-a-days. The North Star plates carry 1 ounce of silver per square foot, those of the Empire have 2½ ounces, while at the W. Y. O. D., the newest plant, the amount has been increased to 5 ounces. One cannot remember too often that it is amalgam that catches gold, and that a good coating of gold-amalgam is better than all the nostrums in creation. In starting a new mill or introducing new plates a good coating of silver helps to get the tables into working order in a short time, the silver being gradually replaced by gold. There is no economy in poorly-plated tables or in short, narrow ones. No mill I have ever seen had too much amalgamating-surface; many of them have had too little. The value absorbed by the plates is an asset of the best kind; and there is no plate placed in a mill but will absorb some gold and be worth more when it is worn out and discarded than when it was first put in place. The following fact* will be of interest in this connection. The old plates of the 60 stamp-mill of the Montana Co., Ltd., at Marysville, Montana, after 4½ years' steady work, 1887–1891, were scraped and melted down. (This was, of course, after they had undergone the usual periodical clean-up.) The 12 plates, each 54 by 96 inches, yielded $90,000. One plate gave as much as $8000. Even the small vanner plates,† 16 by 48 inches, yielded $500 each.

The importance of the careful sampling of the tailings as a check upon the mill-work is better appreciated at Grass Valley than in any other locality I have visited. At the Empire mill there is an automatic sampler, invented by Mr. Starr, the former superintendent, which does excellent service. The results indicate a saving of from 85 to 87 per cent., which is fairly representative of the district.

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* Which I owe to Mr. R. T. Bayliss, the general manager of the Montana Co., Ltd.

† In the neighboring 50-stamp mill.

Notes on the Kaolin- and Clay-Deposits of North Carolina.

BY J. A. HOLMES, STATE GEOLOGIST, CHAPEL HILL, N. C.

(Atlanta Meeting, October, 1895.)

As the Appalachian mountains reach their maximum development in western North Carolina, we find also in that region indications of extensive dynamic disturbances and alterations undergone by the rocks in connection with these mountain uplifts. Among the minor results of these changes have been the formation of numerous dikes or "veins" of exceedingly coarse granitic materials, which in some places are mined for the mica which they contain, and in other places are quarried for kaolin. These dikes are filled with quartz, feldspar, and mica, in varying proportions, very coarsely crystallized. Sometimes the mica (generally muscovite), sometimes the feldspar (generally albite or orthoclase) predominates. When the mica is present in considerable quantity, and in large crystals, the dike is usually spoken of as a mica-vein, and is often worked for mica. Sometimes these crystals of mica occur on one side or the other; sometimes on both sides; and sometimes they are largely concentrated toward the middle of the vein, or, it may be, distributed throughout the dike with a considerable degree of uniformity. In many cases the crystals of mica are too small and few to permit the working of the vein as a mica-mine; in other cases, very little mica is present, and the feldspar constitutes the larger part of the material. This feldspar of the dikes undergoes, through the weathering action of the atmosphere, certain chemical changes, resulting in its alteration from feldspar into kaolinite—the kaolin of commerce.

These dikes vary considerably in size, ranging from a few inches to several hundred feet in thickness, and up to many hundred yards in length. They are generally parallel to the schistosity of the crystalline rocks, which, however, in some cases, they cross at varying angles.

The kaolin in those dikes which occur in the Unaka or

Smoky mountains, is said to have been mined by the Indians, packed across the country to the seaboard and shipped to England, as early as the seventeenth century.* From one of them, near Webster, in Jackson county, kaolin is now mined (by the Harris Clay Co.), and shipped to Trenton, N. J., and other centers of the manufacture of fine pottery. This Webster dike contains very little mica, and comparatively little quartz. It has a maximum width of about 300 feet, and has been traced for a length of more than half a mile. It is mined to a depth of from 60 to 120 feet, below which the original feldspar has not been sufficiently altered, and is too hard for economic mining. The kaolin is brought from the mine, crushed, and washed in a series of settling-vats, for the purpose of separating it from the granular quartz. Its plasticity is increased both by this washing and by the subsequent grinding which it receives. The following analysis† of the washed and dried product, ready for shipment, shows the general character of such material:

Analysis of Kaolin, Harris Mine, near Webster, N. C.

	Per cent.
Free silica, silicic acid, and sand,	2.28
Combined silica,	41.62
Alumina,	40.66
Oxide of iron,	0.14
Alkalies,	0.46
Lime,	none.
Magnesia,	trace.
Combined water,	14.00
Moisture,	.84
Titanic acid,	none.
	100.00

Many somewhat similar, but smaller, feldspathic or kaolin dikes have been found in the various other counties west of the Blue Ridge, and at a number of these the feldspar has been altered into kaolin for considerable depths below the surface, but none of them are now worked extensively for either the feldspar or the kaolin, except the Harris Clay mine just mentioned. Also, at various points in the Piedmont plateau, which extends

* W. C. Kerr, *Trans.*, viii., 1880, p. 462.

† Made for the Harris Clay Co., of Dillsboro, N. C. at the Pittsburgh, Pa., Testing Laboratory.

east of the Blue Ridge for from 150 to 200 miles, there are to be found deposits of this kaolin which have doubtless originated in much the same way as those west of the Blue Ridge; but none of these are now worked to any considerable extent. The age of the crystalline rocks in the Piedmont plateau and the mountain counties, and the exact time at which the disturbance took place which resulted in the formation of these massive granitic dikes, is, as yet, a matter of doubt.

So numerous are these dikes in certain places, and so long have their feldspars been undergoing surface-transformation to residual kaolin or clay, that one might expect to find in this region, as in some other countries, sedimentary deposits of this material, which had been transported for greater or lesser distances; but when we bear in mind the general elevation of the mountain-region, and the consequent rapidity of its streams, we can readily understand that this product of decay and denudation would scarcely be deposited until it had been carried so great a distance from the original source as to be lost by commingling in the lowlands with larger proportions of other and different materials.

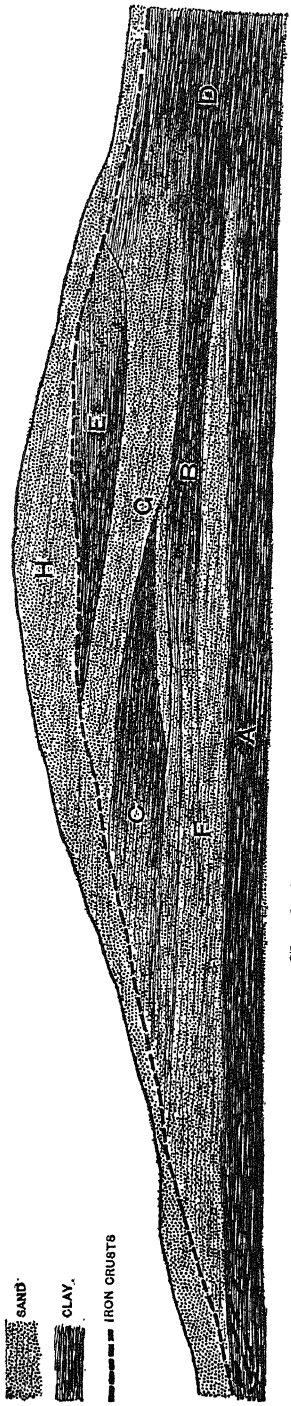
Along the borders of the Piedmont plateau-region there are occasionally found deposits of this kaolin-material, which has evidently been carried but a short distance. Such occurrences are more extensively known on the western border of the Coastal Plain region, mainly in the Potomac formation, as in the neighborhood of Aiken, S. C., and Augusta, Ga., and in many other places, where considerable deposits of this kaolin-material occur, both in the form of arkose (where the kaolin is still mixed with the quartz and mica of the original granitic formation) and in the clay-beds, where it has been more completely sorted, and the kaolin has been separated from the coarser materials, so as to form extensive beds of what is locally termed "china" or "potter's" clay. In some cases, in the arkose material just referred to, the partially decayed crystals of feldspar are frequently found with kaolinization incomplete; and mingled with these are fragments of other minerals, transported from the *débris* of the crystalline rocks occurring along the borders of the Piedmont plateau, not many miles away.

The points above noted may explain, perhaps, the origin of a part of the confusion which has arisen in the use of the term

"kaolin." The applicability of this name to the material described above as having its origin directly in the masses of the feldspar in the large granitic dikes, I suppose no one will question. But if the residual material of dike-decomposition has been transported a short distance by the streams and deposited without further sorting the materials, or if it has been transported to a much greater distance, so that the sorting has become fairly complete, and the mineral kaolinite, while separated from the quartz and mica of the original mass, remains unmixed with other foreign materials, so as to be itself fairly pure,—the question arises, whether the term kaolin is still applicable in both cases, and if so, to what extent, in its transportation and sorting, this material may become mixed with other foreign materials, resulting from the decay of crystalline rocks in the region through which it has been transported, before the term kaolin would become inapplicable. In other words, where, in such a case, should we discontinue the use of the word "kaolin" and apply the broader term "clay." Further discussion of this question cannot be attempted in this paper; but it is mentioned here, because the writer has recently heard a number of complaints from practical potters who use the clay-materials on a commercial scale, that many people throughout the country were designating all the samples of their materials forwarded as "kaolin," regardless of their color, origin and other general characteristics.

Through many places, in both the mountain- and the Piedmont plateau-regions, there are deposits of clay resulting from the decay of the granites, gneisses and crystalline schists. Many of these have a structure which would indicate that the materials have been transported for greater or less distances. But in, perhaps, many other cases, the materials have evidently decayed in place, since the gradations can be traced from the clay down into the partly altered rocks below. These clays, of course, vary in composition with the character of the rocks from which they have been formed. They have frequently a reddish or yellowish color, due to the oxides of iron present, though in many places the colors are much lighter, the iron having been removed through the action of organic matter. In other cases they are bluish or blackish, from the presence of organic matter. As will be seen from the above statement,

FIG. 1.



Clay beds in Prospect Hall Bluff, Cape Fear River.

A, clay 10 feet thick, above medium low water; B, clay 6 to 8 feet thick; C, clay 12 to 15 feet thick; D, clay 40 feet thick; E, clay 15 feet thick; F, laminated sand with thin clay partings; G, sand with thin clay partings, 12 feet thick; H, laminated yellowish and whitish sand with thin loam partings, 15 to 18 feet thick, becoming loose, nearly pure whitish sand near the surface, except near the left end—down the river. The iron crusts are much less prominent than is indicated in the drawing.

these may be classed as partly residual clays and partly transported clays. They have been worked on a small scale in many places for brick; and in a few places, as at Biltmore (Buncombe county) and at Pomona (Guilford county), they have been used in the manfacture of tile, drain- and sewer-pipes; also at Pomona for fire-brick; and near Grover (Gaston county), for fire-brick and vitrified or paving-brick.

The age of these transported clays of the mountain- and Piedmont plateau-counties is unknown. Some of them, upon careful investigation, may be shown to belong to certain definite recent geologic periods; but many of them, probably, cannot be ascribed to any definite geologic time, but must be attributed simply to local conditions; and their age is probably recent. The clay and brick-loam deposits along the river-terraces of the mountain and Piedmont counties, which in many places are well adapted to the manufacture of brick, may be Columbian, or older in age.

Those residual clays of these regions which have been formed *in situ* are the result of the processes of decay, the operation of which cannot be limited to any definite epoch; but these existing deposits may be ascribed, in general, to recent geologic time.

The most extensive beds of clay known in North Carolina, are those found in the Coastal Plain region. In the Potomac (lower Cretaceous) formation there are extensive beds of laminated, dark-colored clays, exposed along the banks of rivers crossing the Coastal Plain region, notably on the Cape Fear river, for 50 miles below Fayetteville. These clays are usually dark in color, owing to the vegetable matters which they contain; and, in some cases, they are highly lignitic. The thin laminæ of clay are frequently separated by still thinner partings of sand; and frequently within a short distance (from a few feet to a few hundred feet) the clay laminæ become thin and disappear, while the sand-partings gradually thicken, so that the whole assumes the character of a sand-bed instead of a clay-bed. This feature, which indicates plainly the shifting conditions under which these deposits were laid down in certain localities, is illustrated in the preceding sketch of the river bluff at Prospect Hall, on the Cape Fear river, 21½ miles below Fayetteville.

In some portions of these clay-beds, pyrite occurs in such quantities as would probably interfere with their industrial use; but the larger portion of the deposits appears to be free from pyrite, and will probably prove to possess considerable economic value. Thus far, no efforts have been made to utilize them; but both analytic and practical tests of them are being made at the present time.

Along the western border of the Coastal Plain region, especially in Moore and Harnett counties, there are limited exposures of siliceous Eocene deposits (overlying the Potomac series, and capping some of the sand-hills), which have recently been tested for fire-brick with very satisfactory results. These deposits are from 5 to 15 feet, or more, in thickness, and are overlain by but a few feet of loose sand. The following analysis* of this material, collected 2 miles northeast of Spout Springs, a station on the Cape Fear and Yadkin Valley R. R., shows its general composition:

Analysis of Fire-Clay (Eocene) Two Miles Northeast of Spout Springs.

SiO_2,	87.70
Al_2O_3,	3.29
Fe_2O_3,	2.81
CaO,	0.48
MgO,	0.40
Alkaline chlorides,	1.48
Loss on ignition,	3.15
	99.31

Among the Miocene deposits there are, in places along the river-bluffs of the Coastal Plain region, especially on the Roanoke and the Tar, somewhat extensive exposures of "blue marl," a calcareous clay, which may prove to be of some value, but of which no practical tests have yet been made.

The Lafayette (Pliocene) materials, which are spread over so large a portion of the Coastal Plain region, are generally gravelly or sandy in composition, with a large admixture of loam in many places. No extensive deposits of clay have been observed among the materials of this formation, though doubtless limited deposits of clay will be discovered as more extensive explorations are made.

* Made in the laboratory of the N. C. Geological Survey. (Analysis No. 354.)

The later deposits, bordering the river-courses and covering the river-terraces, at elevations from 25 to near 100 feet above sea-level, which may be designated as the river-phase of the Columbian formation, contain extensive beds of sandy clays and brick-loams; and, throughout the entire Coastal Plain region of the south Atlantic States, it is these deposits which are most largely used in the manufacture of brick.

Notes on the Underground Supplies of Potable Waters in the South Atlantic Piedmont Plateau.

BY J. A. HOLMES, STATE GEOLOGIST, CHAPEL HILL, N. C.

(Atlanta Meeting, October, 1895.)

IT is a fact that is coming to be more widely recognized by the general public, as well as by members of the medical fraternity, that the health of persons living in our hill-country depends in no small degree upon the drinking water obtained, just as it has been found that the use of pure water in the lowlands and swamp-areas of the Southern States results in practical immunity from malarial diseases. Hence the problem, how to obtain supplies of wholesome water for the towns and manufacturing establishments in the hill-country or Piedmont plateau-region of the Southeastern States, becomes one of considerable interest, the importance of which will continue to increase, as the favorable conditions for manufactures and agriculture in this region will make it, in the near future, the most thickly populated portion of the South Atlantic States.

Water-supplies from surface-streams are unquestionably of the first importance; and in the mountain counties, where the region is still largely forest-covered and the streams are swift and continually aerated by rapids and cascades, the water is of superior purity and clearness. This statement is also applicable to the more elevated and sparsely settled portions of the Piedmont plateau; but in the less hilly and more thickly settled portions of this region the streams are more sluggish, and the waters more muddy and less pure, owing to the fact that a much larger proportion of the surface is under cultivation.

Furthermore, many of the towns and manufacturing establishments are located at distances from the larger rivers and creeks too great to permit of the water being lifted and transported to them by pipe-lines at any reasonable cost.

Rain-water, caught from the roofs of houses under favorable conditions, and kept in properly constructed cisterns, is probably the safest for drinking-purposes; but under unfavorable conditions, and when not properly attended to, cistern-water must be considered as not altogether safe; and in any case the supply is inadequate for large establishments.

Such being the case with regard to surface supplies of water, it will be seen that, in a number of cases, we must depend for potable waters upon underground supplies. These may be obtained either from springs or wells. Of the latter, we may consider three varieties: The ordinary open well, such as is often seen about private residences; deep bored wells which penetrate the crystalline rocks in the endeavor to obtain artesian supplies of water; and the shallow bored wells which are put down through the soil to the surface of these crystalline rocks in the hope of striking underground currents along the lines of contact between the lower portion of the soil and the upper portion of the undecomposed rock. In this latter case, generally, several such wells are bored within a short distance of each other, and these are connected by iron pipes, and water is pumped from the various pipes through a common pipe to a common reservoir or tank. This is what is generally known as the tube-well system.

The open springs furnish an excellent but limited supply of water for family use; a supply, however, which, though sufficient for the needs of isolated residences, is generally inadequate to meet the demand about towns and manufacturing centers. Furthermore, in such latter cases, and frequently even near isolated country residences, the surface in the neighborhood of the spring becomes so contaminated with decaying organic matter that the water becomes unfit for drinking-purposes. The same general statement may be made concerning ordinary open or driven wells, which, for the sake of convenience, must be located near residences where the surface-soil becomes more liable to contamination as the region becomes more thickly settled. Examples of this are not infrequently seen

where the water from wells and springs in newly settled communities is found to be healthful, but a few years later has become so contaminated with organic matter, which has permeated the soil from above, that sickness follows its use, and it must be finally abandoned. It is difficult, however, to get the average citizen to understand that the organic matter of the water in his well or spring may come from the soil immediately about his premises, as the prevailing notion concerning these supplies of water is that they come, not from the immediate vicinity, but from some distant region. Consequently, in many of our towns and even about the isolated country residences, the barnyards and the privies and the hog-pens seem to be built upon the principle of convenience alone, which frequently places them in close proximity to the well or spring from which the family supplies of drinking-water are obtained.

But, outside of this question as to the purity of the water, the supply of water from isolated springs and open wells is generally quite inadequate for towns or manufacturing-establishments of any considerable size, unless the number of these wells is greatly multiplied; and their multiplication means their wider distribution through the settlement or community, and thus a multiplication of the possible sources of disease from the drinking of contaminated waters. Nevertheless, the fact remains that many of the towns of this region, with a population of from a few hundred to several thousand, are still without any general supply of water other than that from independent shallow wells. And while the amount of disease in such cases generally increases with the age of the town, and the physicians, at least, recognize the increasing contamination of the water as the source of this increase in sickness, yet, for the lack of a better system, this one continues in existence.

Deep artesian-well supplies are not to be depended upon, for the reason that the geologic conditions in the Piedmont plateau-region are not favorable. The rocks of this region are crystalline schists, gneiss and granites, with the dips (schistosity) generally steep and varying on both sides of the vertical. A considerable number of borings, varying from 100 to 1000 feet in depth, have been made into these crystalline rocks in the Piedmont region of the two Carolinas and Georgia, during the past few years, with the expectation of securing either an

"artesian" (overflow) supply, or a supply that would come near enough to the surface to be reached by the pumps. But the results have been generally unsatisfactory; the holes being "dry," or the supply of water being inadequate. A somewhat exceptionally favorable result was experienced in Atlanta. Some years ago (1881–82) a well was bored into the gneiss-rock in the heart of Atlanta to a depth of about 2200 feet, at a cost of about $20,000. At 1100 feet a large supply of water was tapped, which rose to within about 17 feet of the surface. For several years this well constituted the water-supply for a considerable part of the city; but the water was pronounced unsafe by the medical authorities, and the well has been abandoned in favor of a water-supply from the Chattahoochee river. In a few other cases exceptionally large supplies of water have been reached; but as a rule, the boring of these wells has failed of satisfactory results.

Some professional well-borers, like some professional miners, with a laudable desire to be kept busy, urge that the deeper the hole the better the chances of success; an opinion that has frequently but slight foundation in the case of the mines, and is, in the case of well-boring in this region of crystalline rocks, contrary to both theory and experience. The possibility of exceptions no one will deny, as we see that, in a few of the deeper mines of this region, considerable streams of water are tapped; and in some cases there is a bare possibility that the hole to be drilled for a water-supply may tap such an underground stream of water, as was the case in Atlanta; but the chances are more than ten to one against such "luck." As a rule, these crystalline rocks become harder and more solid as we descend, and the chances of securing a reasonable supply of water, which are never good after the hole enters the real mass of rock, may be said to decrease as the hole descends. There is, however, one certainty about this operation, namely, other things being equal, the deeper the hole the more rapidly the cost increases.

During the past few years the tube-well system mentioned above has been introduced in a number of communities in the Piedmont region, with decided success in furnishing a good supply of drinking water to the smaller towns and manufacturing establishments. This system is based upon the existence of fairly well-defined underground "currents" of water, in regions

where the topography is favorable, where the rocks have decayed to a considerable depth, and where, near the lower limit of this decay, there is more or less porous material, through which water may readily percolate. Of course, it has been well-known in the past that more or less well-defined underground movements of water existed, that at favorable locations the small currents come to the surface as springs, and that frequently, both on elevated regions and about lowlands, when wells are sunk sufficiently deep into the soil,—usually near the surface of the hard rocks,—a sufficient amount of water is found either to empty into the well as a small stream or to ooze into it from the surrounding soil, and thus furnish a limited supply. But it is only recently that the location and extent of these underground sources of water have been investigated, in some regions with considerable care, and they have been found capable of yielding under proper treatment much larger quantities of water than have been reckoned upon in the past. This investigation has been prosecuted in this region mainly by Mr. Henry E. Knox, Jr., a hydraulic engineer of Charlotte, N. C., who has in this way located considerable supplies of underground water in regions where they were sorely needed.

I give below, in tabulated form, the results obtained by Mr. Knox in the Piedmont region of North and South Carolina. His method of investigation is to examine carefully the topography and geology of the region where the water-supply is needed. The topographic conditions favorable to success are, as might be expected, where there is more or less of a basin, shallow ravine or valley, so that the water which falls upon the surface, instead of running off in opposite directions, naturally percolates downward if the soil is sufficiently porous, and tends to concentrate along the lower portion of such basin or valley, where it may meet with least resistance in the more porous materials.

By way of exploring such a region, a number of holes are bored in line across the basin or valley, so as to determine the existence and location of such an underground "current" of water. In this way, its position at intervals is determined, and the intervening course is traced by additional borings. If the water-supply is tapped by these borings it sometimes overflows; the quantity thus overflowing is measured, and pumps are then

applied so that the possible yield of water can be estimated. In these underground "streams" the water usually follows the topographic conditions, as might be expected; but in some cases it moves more or less obliquely across the ravines, showing that the overlying soil has not the same thickness everywhere, *i.e.*, that the topography of the soil-surface is not the same as the topography of the underlying rock-surface; and that the water-current moves along down the incline of least resistance of the rock-surface, independently, in a measure, of the topography of the soil-surface.

The fact that the water percolates through this more or less porous material at considerable depths below the surface, of course suggests that the movement must be sluggish; but that there is a definite movement is shown by the fact that where there are a number of holes bored at intervals along the line of the "stream," and coloring-matter is introduced into one of them, the color appears in a short time in the water coming from the neighboring holes in one direction, but not in the water from the holes situated in the opposite direction. The average rate of movement, however, has not been determined with a sufficient degree of accuracy to admit of its being stated. These currents are quite limited in their width, ranging in the cases tested, from a few feet to (in rare cases) more than 100 yards. And, as might be expected, the width is not at all constant; but while it gradually increases further down the "stream," as the supply of water becomes greater, yet this increase of width is by no means constant. The depth at which these underground water-currents have been found varies from about 20 to nearly 100 feet, and has been generally less than 50 feet below the surface.

The fact that, in the case of some of these wells, the water overflows at the surface, is due to topographic rather than geologic influences. In some cases, especially at Charlotte, N. C., as mentioned in the table below, the flow from a single well amounts to as much as 10 gallons per minute. Here, as in other places where the overflow is slight—even less than one gallon per minute—the amount of water which can be pumped from such a well is considerably larger. Thus, in the case mentioned at Charlotte (Latta Park), there are several overflowing wells, with an average depth of 42 feet. The maxi-

mum natural flow from one of these wells is 10 gallons per minute, but with the application of a pump the eight wells yield readily 280,000 gallons per day. Again at Chester, S. C., one well, the natural overflow of which is 6 gallons per minute, yields, with the aid of a pump, nearly 62 gallons per minute, or 89,280 gallons per day. In another case, the maximum natural overflow of any one of the eight wells bored at the Western Hospital, at Morganton, N. C., is only 4 gallons per minute, while the eight wells, with an average depth of about 39 feet, yield, upon the application of a pump, 165 gallons per minute, or 237,600 gallons per day.

The quality of the water obtained from these wells has been pronounced satisfactory in every case by the health-officials. Of course, the continuation of this quality will depend largely upon the continued freedom from contaminating influences of these water-basins; and one advantage of this system of water-supply is that the basins, being generally limited in area, may be generally controlled by one or more individuals, or by a corporation, and may thus be kept free from sources of contamination.

As might be expected, the search for underground supplies of water has not by any means been successful in every case; but the limited experience thus far gained leads to the belief that they may be found in a majority of communities, where search is extended over a sufficiently large area, and is made with sufficient care. It would at present, however, be too much to claim that these underground supplies of drinking water can be found sufficient to meet all the demands of larger towns and cities, though they would prove of material service in this connection. But I anticipate that they will prove of greatest importance in connection with the water-supplies of smaller towns and of more or less isolated manufacturing establishments, where there are usually several hundred or several thousand operatives.

The appended table contains a list of the more important places where these underground water-currents have been found, and where the "gang-well" or tube-well system has been introduced; the name of the special establishments for which the wells were bored; the number of wells at each place; the average depth of the wells; the natural overflow per minute from

that one of the series of wells from which the overflow is largest; and the aggregate yield of water in 24 hours from the wells at each place, when the steam pump is applied. The data for this tabular statement have been supplied by Mr. Henry E. Knox, Jr., who bored all of these wells, and who states that out of 23 surveys made by him, only three were unsuccessful in locating the desired quantity and quality of water.

*List of Flowing Wells in the Piedmont Plateau Region of North and South Carolina.**

Place.	Average depth in feet.	Material in which the water was obtained.	Number of wells.	Maximum flow per minute from 1 well.	Pumping capacity, 24 hours.	For whom bored.
				Gals.	Gals.	
Burlington, N. C...	27.5	Bed of gravel.	5	1	20,000	Aurora Cotton Mills.
Morganton, " ...	28.3	Decomp. gneiss.	8	1	150,000	Burke Tanning Co.
" " ...	39.0	" "	8	4	237,600	State Hospital.
" " ...	41.0	" "	5	1	100,000	Deaf and Dumb School.
Rock Hill, S. C....	56.0	" granite.	4	5	72,000	Winthrop N. and I. College.
Fort Mill, "	35.0	" "	4	5	30,000	Fort Mill Man'f. Co.
Pelzer, "	21.0	" gneiss.	4	10	100,000	Pelzer Man'f. Co.
" "	38.0	" "	5	1	100,000	" "
Chester, "	40.3	" granite.	1	6	99,280	Gingham Mills.
" "	42.5	" "	1	2	30,000	" "
" "	72.0	" "	2	1	30,000	Catawba Mills.
Charlotte, N. C.....	42.0	" "	10	10	230,000	E. D. Latta.
" "	51.5	" "	10	2½	200,000	Water Works Co.
Jonesboro, "	31.0	" schist.	1	1½	5,000	Jonesboro Cotton Mills.
Reidsville, "	28.0	" granite.	10	1	100,000	Edna Cotton Mills.

Some Fuel Problems.

BY JOSEPH D. WEEKS, PITTSBURGH, PA.

(Presidential Address at the Atlanta Meeting, October, 1895.)

THE primary problems of civilization are material ones; their answers are writ in fire. When these problems in their higher aspects have pressed for solution, it has been out of the burning bush that the Divine voice has spoken, or in the cloud and smoke that the Divine finger has traced the rules by which these problems shall be solved. It is with the material problems, however, that we, as engineers, have to deal.

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* Bored by Henry E. Knox, Jr., hydraulic engineer, Charlotte, N. C.

It is by fire that these problems are solved. With a marvellous insight into the secrets of power the Greeks made Prometheus, who stole fire from heaven to bestow it upon mortals, the author of civilization. In the words which Æschylus makes him utter as he lay bound on Caucasus, "All arts among the human race are from Prometheus," and it is from the "bright play of fire that all arts spring." In this myth is thinly veiled the origin of the mechanic arts, and is dimly suggested the part fire has played in their development. The myth also foreshadows the triumph of civilization, by the aid of fire, over the forces of nature, for there should one day be invented a flame more potent than lightning, before whose power Jupiter himself "should fall dishonored."

On the part that fire has played in civilization we need not long dwell. For most of the centuries, indeed, up to the day when boiling water told its secret of power to Watts, its use was chiefly its primary one, to heat and light, and not its secondary one as a source of power. Its chemical effect and some of its direct mechanical uses were known, as well as its calorific effects. It cooked the food, but it did not drive the plow nor swing the scythe, nor thresh the grain, nor grind the meal; it smelted the ores, but it did not drive the blast; it heated the iron, but it was the brawny arm of the smith or the force of the falling water that hammered it into shape, while even as in Prometheus's time the land was still traversed "with steeds in cars obedient to the rein," and the "canvas-winged chariots of the mariner" still roamed over the ocean. In a word, while fire gave heat and light, the sources of power through all these centuries were muscle and wind, and falling water.

It is not these primary, but the secondary and even more remote effects of fire that have caused civilization to move with a quicker step. While all the marvellous effects of fire, which have been for ages the possession of humanity still remain as among its greatest endowments, it is as a source of power that fire in the last hundred years has been of such inestimable value. The beginning of its era of power was Watt's invention of the steam-engine, for which the first patent is but one hundred and thirteen years old. The science of thermo-dynamics based on the principle of the conservation of energy, one of the most important advances ever made in scientific

knowledge, was created as late as the years 1845 to 1855. Even what we have learned in the century just closing as to the power of fire is as nothing to the stupendous power of this agent that shall yet be revealed.

It is quite needless to say that as back of this power is heat, and back of the heat fire, so back of the fire is fuel, and that for the civilized world fuel is but another name for mineral or fossil coal, and that for us the fuel question concerns itself chiefly with the possession, production and uses of coal.

There are other fuels than mineral coal, as petroleum and natural gas and the vegetable fuels. Indeed, the fuel question in England at one time, and not many years ago, especially in the iron trade, was a question of charcoal, but for furnishing the world's heat and doing the world's work none of these other fuels cut any figure except in restricted localities.

There are also other sources of power, as light, and wind, and water-flow, and muscle, and were these as willing, as constant and as untiring and tractable servants as heat from fuel, the power that could be derived from these natural forces is so vast that the problem of power would be solved; but there are night and clouds to stop the work of the light; the wind is fitful; it does not always rain, and muscle tires; but night and day, in storm and calm, in dry and wet, and at the end of the longest and weariest day, fire from coal will always answer the demand for power. Nor can we yet call into play that limitless power we call "solar energy," of which we know so much, with which now we can do so little. In our present impotence, coal is mightier to do our work than even this source of all power.

The power problems to-day, then, are fuel problems, and fuel is mineral coal.

The world has a vast store of this mineral fuel—coal. How much, no one knows. But, vast as are these stores, the consumption in certain countries has been so great that nations have affrightedly asked to know how long the supplies would last. In England the question was discussed by such authorities as Sir William Armstrong before the British Association, Mr. Jevons in his work on *The Coal Question*, and in Parliament by John Stuart Mill and Mr. Gladstone. As the result of these discussions, the alarm over the probable exhaus-

tion, in the not distant future, of its coal supplies was so marked that a Royal Commission was organized, who, arguing from several premises, estimated the duration of the supply at various periods, from two hundred and seventy-six years to over twelve hundred years. But the consumption of coal in Great Britain has increased at a rate much in excess of that upon which the lowest estimate of the Royal Commission was based. In 1780, about the time the steam-engine was invented, it was some 6,500,000 gross tons a year. It had arisen to 27,000,000 tons in 1816; to 50,875,000 tons in 1850; to 84,-042,698 tons in 1860; to 112,875,525 tons in 1870; to 146,-969,469 tons in 1880; to 181,614,288 tons in 1890; and to 188,277,525 gross tons (210,870,828 net tons) in 1894. The result of the discussions on the subject of the duration of the coal supply of Great Britain was the conclusion that if the output increases in the same ratio as it has for twenty or thirty years, the coal will be exhausted in a little over a century. These estimates are now regarded as excessive, as it is conceded that there is in each nation a limit to industrial development which, without considering the great economies in the use of fuel, will also limit the expansion of coal production. M. Gruner places this limit for England at 250,000,000 tons, which supposes a mining population of a million miners and a working population of five millions.

In this country the production of coal has been increasing in a much greater ratio than in Great Britain. We cannot go back to the eighteenth century and give figures of production of coal, nor is that necessary in order to indicate how enormous has been the increase in its production and consumption in the United States. At the Tenth Census, 1880, the production of coal in the United States is reported at 71,481,570 net tons; at the Eleventh Census, 1889, it had risen to 141,229,513 net tons, nearly double; and in 1893, according to the report of Mr. E. W. Parker, of the United States Geological Survey, it was 182,352,774 net tons, an increase of more than 2½ times in thirteen years, doubling about every five years.

Similar increases could be shown for the other great coal-producing countries, as Belgium, Germany, Austria, France and Russia. The world's demands for heat and power are increasing marvellously, while the world's supply of coal is a definite

quantity, and it is an evident proposition that with the exhaustion of its coal not only will the power and influence of a nation decline but even its existence may be imperiled.

The fuel problem, therefore, is not only an industrial one but a political and politico-economical one of the greatest importance.

As the amount of coal in the earth's strata is a fixed definite quantity, in discussing the fuel problem we cannot proceed on the assumption of an increase in this quantity. We may discover deposits of which now we have no knowledge; but this does not add to the world's coal supply, but only to our knowledge of where that supply is stored. Some coal may still be forming in peat bogs; but the amount so being formed or that will be permitted to develop into coal will hardly be worth considering. It is evident, therefore, that an increase in the actual quantity of coal stored in the earth is not a factor of the fuel problem.

It is possible, as has been suggested, that other sources of power, and even of heat, such as chemical action and solar energy, especially the latter, may be, and no doubt will be, largely utilized in the future; but these are questions it is not within my purpose to discuss, and the day when these and other sources of power will be largely used will be when the supply of coal is very much reduced and its value very much increased.

As, therefore, we can expect no increase in the amount of the world's supply of coal, the fuel problem is to increase the efficiency of that we have, to make each ton of coal that remains in the hills do a much greater amount of work than a ton now does.

In a word, the fuel problem is to reduce the waste and increase the efficiency of the coal we possess.

This problem divides itself into three distinct parts—that is, problems connected with:

1. The mining of coal and its preparation for market.
2. The use of coal.
3. The products of the coal other than heat.

These problems all concern waste.

First, as to the problems relative to mining the coal and preparing it for market.

It is estimated that as the result of the coal-mining of the last

fifty years not more than 30 per cent. of the coal remaining in the veins worked has on the average been won and laid down at the point of consumption or use. This loss of 70 per cent. does not, of course, include the loss due to geological causes, and especially to erosive agencies that have cut so deeply and so wastefully into the coal originally deposited in the coal-fields. So great is this geological waste that Professor Lesley estimates that not over 1 per cent. of the coal originally deposited in the great Pittsburgh bed still remains, and yet he declares this bed, robbed as it has been, to be the most valuable deposit of coal in the world. In the later years of this half-century the waste has not been so great; but when all things are considered the estimated loss in mining of 70 per cent. of the coal remaining in the strata is not excessive.

As to the causes of this waste and the methods by which it may be reduced but little remains to be said, at least so far as relates to anthracite coal in this country, since the publication of the *Report of the Pennsylvania Commissioners on Waste of Coal Mining*, of which commission our old friend and some time President of this Institute, Eckley B. Coxe, was chairman.

Mr. Coxe points out that there are two classes of waste in coal-mining:

1. That which is absolutely necessary and cannot be avoided.
2. That which may be diminished or done away with.

The unavoidable waste is defined to be that portion of the coal that must be left in the mine for various purposes. It does not follow, however, that what is to-day termed unavoidable waste is absolutely unavoidable. It may be, with our present knowledge, but what this term "unavoidable waste" really means, in many cases, is simply that under present conditions, or under the engineering adopted, it is regarded as most economical to leave this coal in the mine. It is left to maintain the workings, the slopes, shafts, gangways, etc., to support the surface, to make the floor and roof, to keep the water from the lower levels and save the expense of pumping; in a word, to keep the mine safe, and in such condition that it can be worked economically. Conditions, appliances, and methods, may so change, beyond question will so change, that what is economical to-day, may be most wasteful to-morrow. The history of mining shows this.

Just what is the loss from this so-called unavoidable waste in mining at the present time is difficult to ascertain. In some instances, it is not 10 per cent; in others, as high as 50 per cent., or more. This refers to the veins actually worked and to those portions that are worked, and not to those that are in whole or in part regarded as not workable. If this unavoidable waste average 20 per cent. of the amount of coal produced, the loss from this source in the United States, in 1893, was 36,470,-555 net tons, and on the production of Great Britain, in 1894, it was 42,174,166 net tons. If the waste for 50 years is considered, the amount will be enormous.

This unavoidable waste can be reduced. It is being reduced as the results of the application of engineering skill. When the day comes that the near exhaustion of coal will be a thing of to-morrow, and not of a century, it will be found that a waste that is now called unavoidable will then be termed criminal.

The avoidable waste in mining is largely due to:

1. Miscalculations as to the amount of coal that must be left for the pillars, etc.

2. The leaving of large amounts of coal unmined in a vein.

3. Imperfect work on the part of the miner.

The loss of coal from miscalculations or bad engineering of the mine is enormous. Pillars may be too large and the coal wasted; or too small, and the pillars crush and shut off the coal beyond. It is not unusual to leave unmined a part of a vein that is either under or above a slate, and which may not be quite so pure as that mined. The waste from this source is enormous. There are mines in the Pittsburgh region where, with 71½ inches of coal, but 32 inches of clean coal and the bearing-in coal of 4 inches are mined, 36 inches out of 71½ inches; the rest is left untouched, a loss of 35½ inches; practically, one-half of the coal is left in the mine, besides the waste in mining. This custom is not at all uncommon. The miner may do his work very unskilfully in bringing down the coal, in loading, and other ways to which I need but refer at this time.

As Mr. Coxe so admirably points out, in the report above referred to, there is, in connection with the preparation of anthracite a large amount of loss. This is not so great with bituminous coal, but there are culm and slack heaps about

bituminous as well as at anthracite mines. Mr. Coxe estimates that the amount of coal sent to the culm bank in the anthracite region of Pennsylvania, since mining began, has been 35 per cent. of the total production; or, up to the close of 1892, 315,700,000 tons. At certain collieries, from the year 1820 to 1883, 20 per cent. more coal went to the dirt-banks than was marketed, and it was not unusual for an amount equal to 50 to 75 per cent. of total shipments to go to dirt-banks.

In view of all these facts, the statement that on the average during the last 50 years not more than 30 per cent. of the coal in the measures mined has reached the place of consumption is not at all surprising.

How can this waste be avoided?

It cannot be entirely avoided, but it can be still further decreased by just the methods by which it has already been largely reduced. Mechanical means, instead of the coal itself, can be used for supporting the roof and surface; gobbing up will often give a much larger percentage of coal; better engineering of the collieries will give better methods and less waste. All of the vein can be mined, even if a portion of it is inferior, and many methods can be greatly improved.

Secondly, as to the problems connected with the use of coal.

It is estimated that not to exceed 10 per cent. of the possible energy in coal is utilized; indeed, 5 per cent. is the amount most frequently named. Some of this loss is unavoidable, and will ever be so until we have solved that greatest of all modern industrial problems, how to obtain energy direct from coal.

But much of this waste is avoidable, and to reduce this avoidable waste is the fuel problem in connection with the use of coal.

What has been done in reducing this waste in the last 100 years is simply astonishing, and what is of especial interest is that many of the improvements and processes that have been introduced into the arts within this time have been the results of attempts to diminish this waste of fuel. The story of many of these inventions, could they have been told by a Stevenson, would have given us new Arabian Nights tales more marvellous than those he did tell. What marvels have been performed by the genii Watt unloosed from the boiling kettle! The gas Mur-

dock told men how to use is a veritable Aladdin's lamp, and the stories of the wonders of Neilson's hot blast and Bessemer's fiery converter and Siemen's furnace of the heat-cycle and many others are marvellous almost beyond belief.

I do not propose to-night to weary you with the details of what has been done in fuel economies in the arts, but to briefly outline these accomplishments merely as an indication of the line on which future problems may be solved.

Fuel in the arts has at least a three-fold use:

1. As a simple heat agent.
2. As an agent of chemical changes.
3. As a source of power or, better, energy.

We have no complete estimate of the consumption of coal for various purposes in the United States, but the Royal Coal Commission, in 1871, and Mr. Price Williams, in 1889, made very careful estimates of the distribution of coal consumption for Great Britain. I had hoped to obtain an approximate estimate of the consumption of coal for various purposes in the United States, but so far have failed.

The estimate of the Royal Commission is as follows:

CONSUMPTION OF COAL FOR DIFFERENT PURPOSES IN GREAT BRITAIN—COAL COMMISSION ESTIMATE, 1871—TONS OF 2240 POUNDS.

Uses.	Tons.
Iron,	32,446,606
Power and general manufacturing,	25,327,213
Domestic,	18,481,527
Gas and water,	7,811,980
Mining and collieries,	7,225,423
Steam,	3,277,562
Railroads,	2,027,500
Smelting, other than iron,	859,231
Miscellaneous,	195,045
	97,652,087

Percentages for Various Uses.

	Per cent.
Metals and mines,	44
Domestic, including gas and water,	26
General manufacturing,	25
Locomotion by land and sea,	5

CONSUMPTION OF COAL FOR DIFFERENT PURPOSES IN GREAT BRITAIN—WILLIAMS'S ESTIMATE, 1889.

	Per cent.
Production of steam power, including collieries,	30.30
Manufacture of pig-iron and metallurgy,	17.26
Navigation,	8.66
Railways, including fixed engines,	3.98
Water-works and miscellaneous,	1.40
Domestic use,	17.44
Gas manufacture,	5.87
Export,	15.09
	100.00

It is impossible at this time to distribute this consumption even approximately into the three uses, heating, chemical and power, but it will be seen readily that by far the larger consumption is for power and in iron manufacture. It would seem from an inspection of these two estimates that from 60 to 70 per cent. of the coal consumed is for power, and for iron and steel manufacture.

What economies have been wrought in these industries in the past 100 or 150 years?

In engines since the days of the Newcomen engine the duty of 94 pounds of coal has increased from 7,450,000 foot-pounds to 140,000,000 foot-pounds, the highest duty of Mr. Leavitt's pumping engine at Louisville, and the pounds of coal per indicated horse-power decreased from 26.6 pounds to 1.33 pounds, an increase in duty of 20 times, and a reduction in coal, of course, to $\frac{1}{20}$.

The following table shows what has been done in detail:

Duty of Cornish Engines.

	Duty in foot-pounds per bushel (94 pounds) of coal.	Pounds of coal per 1 H. P. (estimated).
Newcomen,	7,000,000	26.6
Smeaton,	10,000,000	18.6
Watt, 1800,	20,000,000	9.3
Lean's report, 1815,	52,300,000	3.6
Lean's report, 1827,	67,000,000	2.8
Lean's report, 1834,	98,000,000	1.9
Lean's report, 1840,	107,000,000	1.7

Highest duty of Mr. Leavitt's pumping engine at Louisville 140,000,000 per 100 pounds of Pocahontas coal. Pounds of coal per indicated horse-power, 1.33.

As to the reduction of waste in iron and steel manufacture: To an audience like this, to show what has been accomplished

I need only mention Neilson's hot-blast, the Bessemer or pneumatic process, the Siemen's regenerator.

One example of the reduction in the use of coal in iron manufacture must suffice. In 1828 Neilson invented the hot-blast, which Mushet said "ranks with the invention of cotton spinning." At the Clyde Iron Works in 1829, for a six months' run, the average coal consumption per ton of pig with cold-blast was 8 tons, 1 cwt., 1 qr.; for six months in 1830 at the same furnace with the same blowing-engine the consumption with hot-blast was 2 tons, 5 cwt., 1 qr. To-day we are making pig-iron with a ton of coke, or 1½ tons of coal to a ton of pig.

This indicates what has been done in the arts in reducing fuel consumption.

But our modern practice has not reached the limits of economy. If we get but 10 per cent. of the available energy out of our coal there must be a vast field for the exercise of engineering skill in reducing this enormous waste of 90 per cent. Sir William Armstrong states that without carrying economy to extreme limits, all the effects now realized from the use of coal could be obtained by an expenditure of half the quantity.

In what direction are we to seek for the answers to the problems connected with the use of coal? I can only briefly indicate them.

They are:

1. A more perfect combustion; that is, from the same amount of fuel more heat units must be developed.

2. Improved appliances for saving this heat and transmuting it into energy. Not only must these increased heat units do more work, but each individual heat unit must directly develop more energy.

3. Recuperation of so-called exhausted energy; that is, the heat must continue at work until the actual limit of exhaustion has been reached.

The use of gases instead of solid fuel is an example of the first direction in which we are to look for the answers to the problems connected with the use of coal. The improvements in the steam-engine noted above are examples of the second class, and the Siemen's regenerator and compound-engines of the third.

Thirdly. The saving of fuel products other than heat.

As has been pointed out, it is as a source of heat and power that fuel, that is, coal, is most useful to the world. But it has been learned that there is locked up in this fuel a marvellous series of products which for their beauty, their wonder, their sweetness, their use, are unexcelled, and the end of the story of these products is not yet.

To indicate what these products are, I need but name the brilliant coal-tar dyes, such contributions of untold value to the healing art as phenacetin and antipyrine, many of our modern perfumes and essences, and saccharine, sweeter than cane-sugar.

While the discovery and character of these products are the fairy tales of science and while the products themselves are of untold value to mankind, there are fuel products other than heat and power and other than these coal tar products that in amount and value far exceed these dyes and medicines and perfumes. The chief of these are the tars themselves from which are derived the light and heavy oils, the creosotes and benzoles, the ammonia from which we get that most valuable of all fertilizers, sulphate of ammonia, upon which the exhausted fields of our country must depend for their renewal of power. From the nitrogen of this fuel, we may also obtain that most poisonous of drugs and yet that valuable agent in gold extraction, cyanide of potassium.

The amount of these products contained in every ton of coal and the consequent amount that we are every year throwing into the air as waste, aggregates an amount almost beyond belief. From every ton of coal coked in the United States, it is fair to assume that in any of the by-product coke ovens, there can be produced at least 3 per cent. of tar worth $\frac{1}{3}$ of a cent a pound; 1 per cent of sulphate of ammonia worth 3 cents a pound; $\frac{1}{2}$ of 1 per cent. of benzole worth 2 cents a pound; and 1 pound of cyanide of potassium worth 50 cents a pound. As in 1893 14,916,147 tons of coal were coked in the United States, the possible production and value at present prices of these products would have been as follows:

Materials.	Amount. Pounds.	Value.
Tar,	596,645,880	$1,988,820
Sulphate of ammonia,	298,322,940	8,949,688
Benzole,	149,161,470	2,983,229
Cyanide of potassium,	14,916,147	7,458,073
		$21,379,810

The above products, however, are only those from the 15,-000,000 tons of coal coked in one year. What about the value of the by-products of the 113,000,000 tons of coal not coked? How many tons of tar and ammonia and benzole and cyanide could be saved from this amount of coal? The amount of ammonia would be something enormous, though the tar and benzole, if the coal was properly burned into gas before it was applied to heating purposes as it should be, would not be so great as when the coal is coked. The Mond circular producer, which I saw at work a year ago in England on Yorkshire coal, gave 48 kilos (105 pounds) of sulphate of ammonia per ton of coal charged, and 80 to 90 pounds was the regular yield, and all this vast amount of sulphate of ammonia, for which our worn-out land all over the United States is crying, is being lost.

Let us recapitulate.

Of the coal still remaining in the seams worked, not more than 30 per cent. on the average in the last 50 years has been won, that is, 70 per cent. has been lost.

The percentage of the possible energy of coal utilized to-day does not exceed 10 per cent., if, indeed, it reaches 5 per cent.

That is, but 10 per cent. of the 30 per cent. of the coal won in our veins, or but 3 per cent. of the possible energy imprisoned in the coal in our hills, is ever released for useful work.

The value of this lost energy must every year, in the United States alone, reach hundreds of millions of dollars.

A low estimate of the value of by-products per ton of coal burned would be fifty cents. This would be $64,000,000 on the bituminous coal mined in 1893. For an age that prates so loudly of its economies, this is a sorry showing.

Is there not in this fuel question a problem that demands most earnest work from our engineers, and in which there is for the miner and manufacturer untold wealth?

DISCUSSIONS.

The Nomenclature of Zinc-Ores.

Discussion of the Paper of Mr. W. R. Ingalls (see p. 17).

(Atlanta Meeting, October, 1895.)

W. P. Blake, New Haven, Conn.: Mr. Ingalls's paper seems to deal more with the synonymy of zinc-bearing mineral species than with zinc-ores. While it may be admitted that there has been, especially in Europe, the confusion of names described by Mr. Ingalls, it appears to me that, under the lead of Dana and his excellent synonymy, we have passed that undesirable condition, at least in the United States, and that there is not now any excuse for confounding smithsonite (the carbonate or "bone" of the miners) with calamine (the hydrous silicate or "silicate" of the miners).

I therefore deprecate the introduction of a new name as proposed for calamine, and think (as Prof. Dana thought and wrote regarding Kenngott's suggested name *hemimorphite*) that such "innovations should have no favor." The proposed name, *hydrowillemite*, is the more objectionable, as there is room for a misconception and misunderstanding of what is meant by willemite. The willemite of Europe, to which the name was originally given, is a very different mineral from the American anhydrous silicate of zinc and manganese, to which the name troostite was given by Shepard long ago. In the desire to generalize and to place under Levy's name, "willemite," all forms of anhydrous silicate of zinc, many mineralogists have lost sight of the fact that our troostite contains a large but variable percentage of manganese oxide, while willemite does not; that the former is peculiarly an American mineral, wholly different in appearance, association, and metallurgical relations from European willemite, and therefore well entitled to the retention of its old name, "troostite," by which it was generally known in New Jersey as late as 1852; and that it should not be subordinated to willemite, nothwithstanding the isomorphism of the protoxides of zinc and manganese.

But for this subordination of troostite to willemite, there would be less objection to Mr. Ingalls's proposition of the name hydrowillemite, which would then apply to willemite only, and not to troostite.

A Water-Cooling Apparatus.

Discussion of the Paper of Mr. Carl Henrich (see p. 43).

(Florida Meeting, March, 1895.)

William Clinton Brown, Brooklyn, N. Y.: The demand for an apparatus for cooling water for condensers, refrigerating-machinery and air-compressors, as well as water-jacketed furnaces, has led manufacturers to put such an apparatus on the market, so that it is now possible to purchase a cooling-tower, properly designed and constructed, by simply stating the amount of water to be cooled and the temperatures of the hot and the cold water.

The principle is similar to that described by Mr. Henrich, but wood is not used in the best apparatus. There are two great objections to wood: First, it cannot last long under the very trying conditions to which it is subjected; and second, it absorbs the organic matter with which water in constant use becomes impregnated, and the resulting odor is very disagreeable.

A few days ago I saw a cooling-tower and condenser, built by the Henry R. Worthington Co., in operation at the Edison electric-lighting station in Brooklyn, N. Y., cooling the water for a 700 horse-power condenser. It consisted of a large stack of iron, about 15 feet in diameter by about 30 feet high. This stack was filled with tiling through which the water falls. A whirligig distributes the water evenly over the top, and a large fan, about 6 feet in diameter, driven by a 10 horse-power electric motor, blows air into the bottom, thus giving a positive circulation. The water, in running down the sides of the tiling, is met by the blast of air, and enough is evaporated to cool the rest. The vacuum-gauge on the condenser showed 26 inches; so the condenser was undoubtedly getting sufficient cool water.

I am sorry Mr. Henrich did not give us the temperatures of the water and the amount evaporated, so that we might compare the two machines. The apparatus which I saw was cooling about 1200 gallons per minute from 112 to 50 degrees, with an evaporation of less than 4 per cent.

The cooling-capacity of a tower depends upon the amount of cooling-surface, the amount of air circulated, the temperature of the air and the humidity of the air. Just what ratio of these is best for given conditions forms the basis for a very pretty little problem; and I would not attempt to give a general opinion.

Nickel and Nickel-Steel.

Discussion of the Paper of Mr. F. L. Sperry (see p. 51).

(Florida Meeting, March, 1895.)

JOHN BIRKINBINE, Philadelphia, Pa.: Mr. Sperry's paper is certainly a valuable addition to the literature upon alloys with iron, supplementing the data already published concerning the influence of manganese, chromium, tungsten, titanium, aluminium, etc.; and it is to be hoped that the presentation of information upon the influences of nickel may bring out comparisons with other alloys.

Taking into consideration the properties of nickel, and its use as an alloy in heavy forgings, as set forth in the paper, there would appear to be no serious difficulties in utilizing this alloy by extending its application to pieces of more moderate size. The melting-temperature of nickel, as stated by Mr. Sperry, is considerably lower than those given by various authorities. Its point of fusion being so near that of iron, and its density but little greater, the difficulty of alloying it in the foundry should not be serious, particularly if it is applied as a ferro-nickel.

The addition of nickel monoxide (NiO) to molten steel would appear to be less desirable than the application of metallic nickel or of ferro-nickel, if such is produced on a commercial scale and at a satisfactory price.

The fact that nickel-coated iron has been in use for years, and the possibility of utilizing it for stamped ware, etc., may offer a medium for the manufacture of vessels to replace some of the glass now employed for transporting table-preparations put up in acetic or other acids. Whether it can compete with sheet-iron or steel, tinned or galvanized, for roofing and building purposes, still depends largely upon the price at which nickel can be offered.

The statements of Mr. Sperry and others concerning the effects which nickel produces upon steel, suggest queries as to the influence which the addition of this metal to cast-iron, either as white iron with the carbon combined, or as gray iron with the carbon largely graphitic, would exert. If carbon reduces the melting-point of nickel, and if, as Dr. Wedding says, nickel absorbs 9 per cent. of carbon, whether in the amorphous or graphitic state; and if nickel replaces silicon in iron as claimed by Drs. Green and Wahl, the addition of varying percentages of nickel to cast-iron may work radical changes in some features of foundry-practice.

Some of the questions which suggest themselves for solution in this connection are:

1. Will the addition of nickel to soft iron castings assist in adapting them to purposes for which chilled castings are used?

2. Can the affinity of nickel for carbon be utilized to transform ordinary cast-iron into malleable cast-iron?

3. At what point will the proportion of nickel added to cast-iron give it maximum hardness? (It is probable that nickel will harden the metal with which it is alloyed up to a certain point, after which it may soften it.)

4. How will the addition of nickel affect the fluidity and the porosity of iron castings?

5. How will the shrinkage, fracture, and other physical properties be affected?

6. Will nickel-additions improve the surface of castings, or cause them to be sandy or scabby?

The results of experiments in the directions named will largely influence the practical problem of using nickel as an alloy with iron in castings, and assist in determining the extent to which, as a higher-priced material, it can be employed.

Unfortunately, the price of nickel, although much reduced from former quotations, makes it still an expensive metal; but as its use is extended there will undoubtedly be economies in production, similar to those which have recently brought aluminium into more general application.

The present price of nickel is given by Mr. Sperry as "below 40 cents" per pound; but it is reported that, at a forced sale in Europe, some nickel was sold at 20 cents per pound. Taking a medium between these, say 30 cents per pound, if 5 per cent. of nickel is added to iron for castings, the cost will be increased 1.5 cents per pound over the cost of ordinary castings; if 10 per cent. is added, the additional cost will be 3 cents per pound, etc. But if the strength is increased by the addition of nickel, and the weight of metal proportionately decreased, the actual cost of adding nickel per pound of castings will be reduced. If nickel added to iron prevents corrosion, the extra cost of the castings would not count as against the permanence of equipment, as in condensers, pipes, standing rigging on vessels or iron piles, particularly if in salt water, etc. To judge of the actual addition in cost, an empirical calculation may be made on the assumption that 5 per cent. of nickel increases the strength of a casting 10 per cent., that 10 per cent. increases it 20 per cent., 15 per cent adds 30 per cent., and 20 per cent. increases the strength 40 per cent., permitting in each instance a corresponding reduction in weight. These proportions are below those given by Mr. Sperry for the increased strength of steel obtained by adding given percentages of nickel, and below what a calculation based on the tensile strength of nickel and cast-iron shows.

Assuming, then, that ordinary castings cost $1\frac{1}{2}$ cents per pound, and nickel 30 cents per pound, we have the following:

	Cents per pound.
Ordinary castings,	1.500
Castings with 5 per cent. of nickel,	2.66
" 10 " "	3.61
" 15 " "	4.447
" 20 " "	5.148

A practical application of an alloy of iron possessing, in a greater or less degree, such properties as are claimed to result from an addition of nickel to steel, would seem to be to replace costly forgings in structural work, particularly those which are

required at points where beams, channels, etc., meet each other at different angles. For such work the additional cost due to the nickel would be of small account.

Mr. Sperry's paper suggests possibilities no more surprising than those which have followed the alloys of other metals with iron and steel; it emphasizes the influences which alloys may exert upon the designs for structural work and machinery, and upon mill- and foundry-practice, and opens a wide field for speculation as to the processes which still dominate our industries; in short, it brings forward the query, "Is this not the era of alloys?"

The reference made by Mr. Sperry to the presence of nickel in meteorites may be supplemented by the statement that the metal is found in varying quantities in meteorites and meteoric stones. In some of the latter, nickel is not reported; but I am not aware of any analysis of meteoric iron in which nickel was sought for and not found, the proportion ranging from 1 or 2 per cent. to as much as 60 per cent. in one specimen reported from Brazil. One specimen taken from an Indian grave in Louisiana, and believed to be of meteoric origin, showed 34 per cent. of nickel; and other masses of well-authenticated meteoric iron, which were seen to fall in the United States, have been found to carry from 6 to 12 per cent. of nickel.

The Physics of Cast-Iron.

Discussion of the Paper of Mr. William R. Webster (see p. 84).

(Florida Meeting, March, 1895.)

F. E. Thompson, Pottstown, Pa.: If Mr. Webster's endeavor to open up the subject of cast-iron should prove as prolific of results as did the discussion on "The Physics of Steel," he must certainly feel richly repaid. His remarks concerning the secretiveness and conservatism of the average foundry-man recall to my mind an instance of a blast-furnace which, for the last year, has been producing much foundry-iron.

The manager, believing that much more of his product could be used in the foundry than the foundry-men were will-

ing to accept under their cut-and-dried tests of fracture and surface, employed a chemist specially to analyze all his output and calculate mixtures for the foundries. He then induced several of his customers to try these mixtures, analyzed the resulting castings for the foundry-men, and eventually made agreements with them that, so long as the castings were satisfactory, no questions were to be asked about the iron. He now naturally finds himself able, under these terms, to dispose of much more of his product as foundry-iron than when the foundry-men did the selecting.

In an article on the subject of applied pyrometry, published in the *Iron Age* of February 21, 1895, I showed that considerable differences in temperature might exist in the same ladleful of cupola-melted iron. I give below some figures showing to what extent the temperature of melted pig may be influenced by the mixture, the burden and other conditions of the cupola. The subject of cupola-practice, Bessemer and foundry, has not been illumined by any great number of papers. A discussion of this branch of Mr. Webster's general scheme would certainly be welcomed by metallurgists and foundry-men.

My figures were obtained principally on Bessemer iron, remelted in the cupola; and any conclusions arrived at might fairly be applied to remelted foundry-iron, so long as the differences in chemical composition did not operate to destroy the analogy. The temperatures were read by the Mesuré and Nouel pyrometer, described in the article above referred to.

The coldest Bessemer iron thus tested was pasty and poured in lumps, being light cherry-red in color and showing a temperature of 951° C. This iron had stood some time in the ladle after being tapped from the cupola, and when poured into the converter was thoroughly chilled.

The lowest reading obtained on iron tapped direct from the cupola was 1034° C., the color being between light cherry-red and orange-yellow.

The hottest iron poured from the ladle into the converter showed 1140° C., and the hottest tapped from the cupola, 1240° C.

The figures show a range in temperature for iron, as tapped, from 1034° C. to 1240° C., and a range in ladle-iron, after standing, from 951° C. to 1140° C.

The greatest drop in temperature occurring while the iron stood in the ladle, preparatory to being poured into the converter, was from 1180° C. to 983° C., or 197° C. The average drop in temperature for 10 tons of iron standing about 15 minutes in the ladle was from 1140° C. to 1068° C., or 72° C.

Foundry-iron, tapped from the blast-furnace, shows a temperature of 1467° C., reaching 1500° C. towards the end of the cast. We have, therefore, a range in molten cast-iron from 1500° C. to 950° C. With a pyrometer capable of reading accurately to 25° C. we should be able to distinguish 22 different points between these limits. Some style of optical pyrometer would doubtless be valuable for such determinations. Of course it must be remembered that the practical limits in the foundry are much narrower, as cold, thick iron at 1000° C. would have to be scrapped, and it would be impossible to obtain temperatures of 1500° C. by remelting in a cupola, even if such extremely hot iron should make good castings.

I give now a few figures showing the effect of changed cupola-burden and of scrap upon the temperature of Bessemer iron. The cupolas in question were being run on a burden of 1000 pounds of coke to 7500 pounds of metal. The iron thus melted had an average temperature of 1140° C. An accessory blast-attachment was put on the cupolas shortly afterwards, and the burden was allowed to remain for a time at 1 to 7½. The iron increased in average temperature to 1260° C. The burden was then changed to an average of 1 to 11 (that is, 1000 pounds of coke melted 11,000 pounds of metal); but the metal dropped in temperature to 1160° C. With the accessory blast-attachment I have melted 16,000 pounds of metal with 1000 pounds of coke for a limited time. The iron was hot enough to blow satisfactorily, but skulled somewhat in the cupola-ladle.

In cupolas provided with the accessory blast-attachment, a very satisfactory run may be maintained for Bessemer work by using a burden of 1000 pounds of coke to 12,000 pounds of metal. I doubt, however, if the iron produced would be satisfactory for foundry-work, being probably too cold.

Working without the accessory blast-attachment, and on a burden of 1000 pounds of coke to 7500 pounds of metal, I find the temperature effect of using mixed iron and steel scrap on the Bessemer cupola to be as follows:

Pig. Per cent.	Scrap. Per cent.		
100		produces iron at	1200° C.
93	7	" "	1180° C.
80	20	" "	1160° C.
70	30	" "	1120° C.
67	33	" "	1120° C.

In remelting foundry-iron, it must be remembered that the smaller quanty of impurities present in the iron would raise the melting-point. This would probably make it necessary to run hotter iron to obtain the same degree of fluidity as is secured in remelted Bessemer iron; hence, a workable foundry-iron would probably have to run in temperature between 1200° C. and 1350° C.

S. M. Vauclain, Baldwin Locomotive Works, Philadelphia, Pa.: In our former practice, so far as making iron castings was concerned, the matter of mixtures was left entirely to the experience of the foreman of the foundry, an able and experienced man, well up in his profession as a mechanic and practical founder, but without knowledge of the scientific manipulation of metals as practiced by some of the best foundry-foremen of the present day. It became apparent, however, that the day for ordinary rule-of-thumb measures had passed, and that more scientific methods should be adopted, not only for the management of the metals after they were delivered to the foundry but in their purchase; and consequently, about six years ago, we scheduled the mixtures we desired for different sizes and weights of castings, practical experience having proved the desirability of different irons for varying thicknesses of castings. To determine the strength of these several mixtures, we made each day two tests of each one employed, and thus kept a partial record of the efficiency of our irons. To show how important this little departure proved to us, I may say that shortly thereafter we were compelled to make castings for one hundred locomotives, to meet a specification of 25,000 pounds tensile strength, each casting to have its own test-piece. This specification is sufficient to indicate the difficulties we should have encountered had we not been experimenting on this line for some time, so as to be in a position to satisfy the most incredulous.

After such an experience as this, we concluded to go into the matter more scientifically and widely, with a view to tech-

nical economies. In handling several grades of cast-iron, products of different furnaces, we had been, at times, at a loss to know "what the cake would be until after it was baked;" and sometimes it was not all that we desired. We have therefore recently called the laboratory to our aid, taking the precaution to have our foreman the central and most interesting figure in the inquiry. Some astonishing results have followed. We have found that the operating expenses can be diminished and the iron improved in quality by proper watching, not only of the nature of the metal, but also of the common workman who throws it into the cupola. Indeed, so much has been shown to depend on the manner of charging, that we are scarcely prepared as yet to give any reliable data of a chemical nature. We have, however, gone sufficiently far to be satisfied that it is a serious mistake to operate a foundry at this date without the aid of an analytical chemist and a foreman who is in hearty sympathy with that feature of the management. We have found also the desirability of purchasing our irons to a chemical specification. At present we specify silicon only; and we shall shortly have a complete specification in shape to issue. As it will be a novelty and subject to criticism, we wish to be sure of our ground before making it public. We have also determined that in handling irons of different chemical composition, we must arrange the metals in the stack in accordance with the result we wish to obtain, and that the metal never comes out in exactly the order in which it is charged, the explanation being that the various irons used have different melting-points. Our practice at present is to cast a test-piece from each tap, then make physical tests of them, and then analyze them. By comparing the strength with the analysis, we can calculate what chemical means we need to produce desired results.

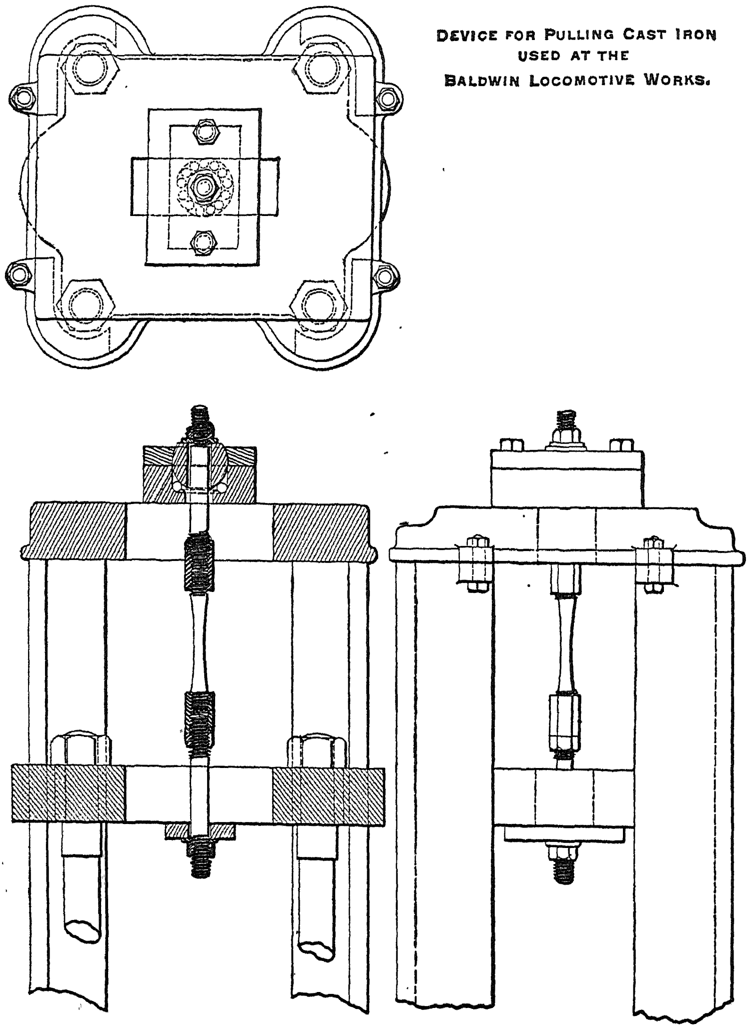
So far as we have gone, we have found it possible to regulate the sulphur. We can readily expel it during the melting, not so far as to eliminate it entirely, but so as to keep it within reasonable limits. We are also of the opinion that sulphur is not so undesirable as generally supposed in cast-iron, except when in certain combinations with other impurities; and from data we have collected it would seem that some of the gun-founders are likewise of this opinion.

We can also regulate the chemical nature of the castings by mixing certain grades, and be perfectly sure of the quality of the casting when made. It must be apparent to any experienced person that the economies thus effected are more than enough to defray the total expenses of operating our entire physical and chemical laboratory.

I do not care to say anything at present with regard to carbon, silicon, manganese and phosphorus, preferring to do so at some future time, when we shall have our diagrams complete, showing the relation between the chemical composition and strength of our castings.

H. V. Wille, Philadelphia, Pa.: The elaboration of a system of testing cast-iron at the Baldwin Locomotive Works has been steadily though slowly progressing since 1876. At that time and for about one year thereafter the quality of the metal was judged by an inspection of the fracture of a specimen cast 6 by 3 by $\frac{5}{8}$ inches in size. Three of these pieces were cast daily; two were broken, and the working qualities of the metal were determined by planing the third one. In 1877, a testing-machine was first utilized for this kind of work. Pieces were cast flat, $1\frac{1}{4}$ inches in diameter, with dates on the sides. These were pulled in the rough; shrinkage- and chill-tests were regularly taken; and all new mixtures were regulated by these tests. No change was made in these methods, with the exception of casting the specimen on end, instead of flat, until 1892. We then became aware that we were making a much higher grade of cast-iron than was shown by our records; and we were soon convinced that there was something radically wrong with our method of testing. In the attempt to remedy this, we first turned our specimens throughout their entire length, and pulled them with the regular Olsen ball-and-socket wedges. As this proved unsatisfactory, we successively tried grooving, turning in 4-inch section and in 4-inch taper-section, without any material gain in tensile strength, the specimens being in all cases pulled in wedges. The trouble was clearly due to a lateral stress thrown upon the specimen so that the pull was not axial. The taper-section was therefore retained, but the ends of the specimens were threaded and pulled in the apparatus shown in the accompanying sketch. The test-piece is screwed into the socket, and always hangs central by virtue of gravity, the friction being minimized by the use of ball-bearings.

Stanley G. Flagg, Jr., Philadelphia, Pa.: Analytical chemistry has before it a large but as yet an undeveloped field in the study of cast-iron; and we have as yet shown nothing that is not common to most foundries making the least pretension to the



DEVICE FOR PULLING CAST IRON USED AT THE BALDWIN LOCOMOTIVE WORKS.

use of chemical formulæ. I do not feel that I am able to contribute anything of value to the common store of knowledge already attained in this direction.

That a full, ultimate chemical analysis of cast-iron will determine its value physically, has, so far as I am informed, never

been shown. That chemistry can be, and is, profitably employed, there is no doubt; but the hope of making it available for the use of small foundries seems almost unattainable.

The knowledge already held to be correct concerning the influences of the different elements upon cast-iron, we use to advantage; and whenever able to get an analysis of our iron we do so, and are guided by these records; but as yet our chief reliance is upon physical tests, with due regard to the chemical influences mentioned.

The only way, in our opinion, to throw light upon this most vital question is by showing the results of long-continued experience in a foundry running chemical analysis alongside of physical tests. This is being done, to our knowledge; and if the management of such an establishment can be induced to make public these facts—though given bare and without any comments or explanations, I believe those skilled in metallurgy can develop laws from them that would be of immense value.

C. R. Baird & Co., Philadelphia, Pa.: The subject of the value of chemical analyses in foundry work has been attracting great attention for some time, and it is conceded by all who have studied it that analyses are of the greatest assistance in making mixtures for all kinds of castings. The best results have been obtained in the most difficult work by following the rules of chemistry in cases where the results would be widely different and quite unsatisfactory, if the irons had been combined according to the appearance of the fracture only. In many modern plants a well-equipped laboratory is considered essential; and that founders also realize the importance of knowing just what they are using is demonstrated by continual requests for analyses of shipments. Only a few years ago furnace-men, as a rule, gave this matter little attention; but now, out of the twenty-four furnaces whose output we handle, eleven maintain well-equipped laboratories, so that they are in a position to furnish a guaranteed analysis of each shipment, if desired.

It is hard for even an expert to judge correctly the character of any iron from its appearance; and consequently it is always necessary to make tests of any new grade or brand before it is possible to reach a conclusion as to whether the iron would answer for the purpose intended. Most founders are business

men and are not disposed to go into the chemical question, feeling that to understand chemistry too much time must be given to its study; and consequently they go ahead on the old "cut-and-try" plan that was in vogue twenty years ago. While this reasoning is true to a certain extent, the slight knowledge needed of the most important elements in an iron might be easily acquired, and we believe would prove of immense advantage and, in many cases, would enable consumers to use cheaper irons with equally good results.

THOMAS D. WEST, Sharpsville, Pa.: In a year or two more, chemistry will have made such an advance that foundry-men will be loth to admit that they ever were guided wholly by fracture. I know there are many, if not the majority, of them who still believe they can judge of iron by fracture. It is only when such persons are placed in a position where they have to contrast the two methods that the necessity of possessing a knowledge of the chemical properties is acknowledged. The writer's experience in working according to analysis has proved the necessity for such a course whenever it is desired to obtain definite and uniform results from mixtures. It is no wonder that foundry-men have experienced so much trouble and expense in obtaining the character of iron desired in their castings. It would be difficult to find a purchaser who buys his raw material with less knowledge of its true qualities than the founder who buys pig-metal wholly by fracture. If he desires a strong iron, he is guided by the amount of sledging required to break a pig; if a soft iron, by the size and color of the grain—two qualities which, when pig is remelted, may be wholly changed. To be assured of obtaining the desired results in every heat a knowledge of the chemical properties before the iron is charged is essential. To know what the chemical constituents have effected, and whether any change in mixture should be made, physical tests are necessary. The writer has been deeply interested in Mr. Webster's late papers illustrating an elaborate series of experiments in estimating the ultimate strength of steel from its chemical composition, and he cannot perceive why, if steel is so sensitively affected in character by variations in its chemical composition, the physical properties of cast-iron may not be largely estimated from chemical analysis. In our foundry-practice we wholly ignore

the judging of pig-metal by fracture, and our physical tests show that by closely watching the sulphur, silicon, manganese and phosphorus (paying the greatest attention to the first two elements) and maintaining uniform conditions of blast-pressure, fuel, etc., in the cupola, we can, from estimated chemical properties, obtain whatever we desire in the physical properties of our castings, with an assurance very far surpassing anything that can be predicated upon guessing at the properties of pig-metal by its fracture.

It is only about one year since the chemistry of cast-iron began to command the general attention of founders, and the rapidity with which it is coming to be regarded as a valuable auxiliary to the foundry is highly commendable. For progress in this direction we need now have little concern. The question most requiring attention is the adoption of some standard for testing the relative natural physical qualities of cast-iron. We urgently require something to record truly the results of changes due to variations in the chemical composition of mixtures, etc.

The physical property of first importance to be defined is that of the strength of an iron; next to this are the qualities of chill and contraction. A round bar, cast on end and arranged so that the test-bar and its "chill" will remain in contact, is well known to be a plan advocated by the writer. The first necessity is, in the writer's estimation, to find out, if possible, what system for physical tests will best record a test to conform with what is commercially known as degrees in the strength of cast-iron. Several months back I presented a paper before the Western Foundrymen's Asssociation giving a series of tests in gun metal, chilled roll-irons, car-wheels, heavy machinery, light machinery, stove-plate and sash-weight mixtures. So far as I know, I was the first to present such a series of tests, and I am pleased to note that the American Society of Mechanical Engineers has recently commenced collecting tests from these specialities. It is not practical simply to collect a series of tests from one specialty and then expect to draw from them deductions which will truly represent the physical properties of cast-iron, as has been the practice of past experimenters. We are to-day in possession of but little information regarding the true characteristic properties and phenomena existing in cast-iron, and it remains for a correct system of physical tests to record and assist in fathoming the truth.

Having once decided upon a correct method for physical tests, all should unite in its use. To understand each other and make comparisons of the physical properties of cast-iron, we all need to use one system. How are we to make comparison or intelligently describe to one another the true physical properties of our tests when one is using a test-bar ½ inch square, another 1 inch square, another 1 by 1½ inches, another 2 inches square, and some 1⅛ inches in diameter, and perhaps no two using the same length of bar? I note with pleasure that the mining engineers are interested in this work, and I trust they will not stop at being interested, but will contribute an active co-operation in research.

William C. Henderson, Philadelphia, Pa.: That the physical behavior of cast-iron is dependent, to a considerable extent, upon its chemical composition is a fact now generally admitted. In other words, the response obtained from a material in service is largely regulated by what that material is. That it is not wholly so dependent is also a fact, equally acknowledged, and one which I think indicates but little progress in the methods of its manufacture.

In the production of steel the disturbing conditions incident to its earlier stages of development, and which are quite similar to those under which the iron-casting is completed, are, of necessity, modified to a practically universal standard, by reheating and rolling; besides which, the chemical composition of steel is generally effected at the very last moment of its fluidity, that is, at a time which precludes, as nearly as possible, the opportunity for change. I believe it to be entirely due to this practically universal standard of treatment that the absolute relationship between chemical composition and physical behavior under such treatment, has been established in the manufacture of steel.

In cast-iron we have finished, at what is but the first step in steel-making, a material which during its brief transition has been made the sport of everything with which it has come in contact. We have its supposed chemical composition imparted to it at the very start of its being, in a melting contrivance in which no attempt is made at controlling the ever-varying conditions of its manufacture. Notwithstanding all this, science has done much, and, of course, will do far more toward the intelligent conducting of such foundries as invoke its aid.

But, while the relation between the chemical and physical natures of the cast-iron made at one foundry may be satisfactorily established, it has been my experience that it by no means follows that such is the absolute relationship, or that such results would apply to the cast-iron made at another foundry, where the conditions of manufacture are totally different. And never, in my opinion, will that absolute relationship be established, except through some such universal standard of treatment as is practiced in the making of steel.

I believe that we should obtain some information regarding the effect of chemical composition upon physical behavior of cast-iron, of immense importance to those at work upon this subject, were a series of experiments made upon test-pieces, all of which were of the same length, had the same depth of sinking-head, and had received treatment in every case identical. The moulds used should be of definite composition and density, to surround the test-pieces proportionately and to receive the fluid metal at a temperature not less than that of the metal itself at the time of pouring, after which the whole should remain undisturbed until the temperature had sunk to that of the atmosphere.

F. Schumann, Tacony Iron and Metal Company, Philadelphia, Pa.: In compliance with Mr. Webster's suggestion I will attempt to give the views of a practical foundry-man, who is assumed to possess but a crude and popular knowledge of metallurgical chemistry; but who, from necessity, is fully impressed with the importance of the value of that science in its bearing upon any advances that may be made in his industry.

Up to within a very recent period but slight advances had been made in foundry-practice. The methods were almost the same as those of many years ago. Except in the greater extent of operations and the gradual formulating of a few empirical rules and improvements in cupolas, fans and hoisting-appliances, no progress could be claimed. Cupolas were known fifty years ago that melted iron just as economically as those of to-day. Blowers and wind-machines were also in existence that delivered all the air required just as efficiently as those of to-day. It is true that such comparatively efficient appliances were not as generally known as they are now, nevertheless they existed. Foundry-men were in practice fifty years ago who knew just as

much of moulding and making castings as we do to-day. We marvel at the strength of some of the old castings, and the skill displayed in their production. Indeed, they knew comparatively more than we do; for they wrought without the aid of the science which we could, but do not, exploit.

Whether the divergence of the industry into specialties such as the making of pipe, car-wheels, malleable castings, etc., has tended to a technical advance, is an open question, although a diminished cost of production has undoubtedly resulted.

It is strange that while blast-furnaces for the production of pig-iron are to-day the very outgrowth of science, having been almost completely revolutionized thereby in recent years, our foundries, so closely related to them, have remained as they were. But dawn is breaking, and the near future will undoubtedly place this industry in line with the general progress of the period.

In no industry is the knowledge of cause and effect regarding a commercial product of greater importance or less understood than in ours. That no systematic investigation in this direction has ever been attempted, is due partly to a skeptical tendency among founders concerning the value of such investigation, and partly to the fault of the chemist, who could give the constituent elements in an iron, but could not give the physical result, which was really what the foundry-man wanted to know.

Any inquiry which results in determining a relation between analysis and physical properties will be, when once established, a factor of vital importance towards our advancement, enabling us to maintain the field against all competitors. Whether the final result will approach the product of the modern steel-foundry or "mitis" metal, is immaterial.

The magnitude and benefit to mankind of the project outlined by Mr. Webster is such that all technical associations, whether of civil or mechanical engineers, foundry-men, boards of trade, manufacturers, etc., should unite in influencing the government to create a commission and appropriate moneys to conduct such an investigation to a conclusion. We should not be the first to do so. Germany has already set such an undertaking in motion.

The investigator will meet with many apparently arbitrary

disturbing causes that will upset his deductions. A few of the effects observed in foundry-practice are stated below. In some cases the causes are well understood; in others, but imperfectly, or not at all.

1. *Fuel.*—The quality of the fuel has great influence upon the casting. Good fuel makes fluid, hot iron, prevents hardness, tends to clean castings, and aids in keeping the cupola in good working-order during the heat.

2. *Iron-Mixture for the Melt.*—In this the most important subdivision of the operation, the smallest advances have been made. Such as are alleged to have been made are not generally known, but are confined to a few foundries or their experts.

What little knowledge we have as to the effects of the constituents upon the physical results, is applied in a crude way. Really, the old "rule-of-thumb" method still prevails in the great majority of foundries. We buy certain brands of iron having a reputation for either strength, hardness or softness, chilling or non-chilling qualities, their capacity to absorb scrap, or their power to resist repeated re-heating without failures (as for retorts, steel-moulds, etc.). We know that silicon induces softness at the expense of strength, and that manganese tends to fluidity, but also produces dirty or porous castings. We also know that repeated re-melting causes hardness, and therefore we avoid an excessive use of scrap or gates and sprues if we want soft castings. We likewise know that sulphur and phosphorus tend to chill or hardness. Upon all these points we have but indefinite knowledge, and therefore are obliged simply to try a mixture in accordance with our notions. If the experiment is successful, well and good; if not, we try again by changing the proportions of the various brands.

The failures are fewer than one would suppose, because we reason that all brands of iron have both good and bad qualities, and that the general average must be good; hence we mix as many brands as possible, and thus obtain what may be fairly called a "neutral mixture." Foreign matter, as usually found adhering to the external surfaces of pig-iron or scrap, whether rust or sand or earth, does not affect the quality of the casting, but does affect the efficient working of the cupola, as it requires more heat and increases the refuse in the cupola and on top of the fluid iron in the ladles.

3. *Fluxes.*—Fluxes are most useful to unite and bring to the surface of the melted iron many of the impurities, and thus aid in cleansing the iron. Oyster-shells are most commonly used for this purpose.

4. *Manner of Melting.*—Variableness in the volume and pressure of the blast has important effects. A cupola may be capable of melting 12 pounds of iron per pound of fuel used, but the result would be hard, chilled castings, while the same mixture, melted at the rate of say 8 pounds of iron to 1 of coal, might make good, soft castings suitable for machinery. Increasing the pressure of the blast, yet still keeping the proportion of 8 to 1, may also result in hard castings.

5. *Casting-Temperatures, etc.*—The temperature of the melted iron, when poured in the mould, and the condition of the mould as to moisture, are of great importance in varying the quality of the casting. Density is a factor of strength. Hot metal induces density, as do also dry moulds, which tend towards slow and uniform cooling.

Moist or green-sand moulds, especially any variation in the volume of sand in the mould surrounding the casting, affects the result, and more sand on one side of the mould will cause a difference in the time of the cooling of the respective sides, and a consequent difference of density and homogeneity, resulting in initial stresses of greater or smaller degree, and affecting the strength of the casting.

6. *Method of Pouring.*—The process of pouring a casting is important. Whether the metal enters the mould at the top or bottom, the rapidity of pouring and flowing in should be so regulated that the accumulating gases will be forced up and out of the mould through the vents. The inflowing currents of metal should be so disposed that they will unite quickly and readily, and tend to the rise of the metal as a body, and thus avoid the flowing together of certain currents that tend to force down facing or impurities ahead of them. These impurities form a film of a lower temperature than the iron, prevent a close knitting or welding of the intersecting currents, and result in co-called "cold shuts." The ferro-static head also tends to an increase of density. The quality of the facing and porosity of the mould greatly affect the density of the casting. Gases may be formed to such an extent that they are enveloped

within the metal in the form of bubbles, either on, or immediately under the face, or in the body of the casting, or by their passage through the fluid metal.

In some cases these defects are solely due to imperfectly vented moulds or gas-producing facing; while in others, they are due to the brands of iron used. Unskilled use of facing may cause seams or "cold shuts," as already pointed out, by interfering with the fusion of intercepting currents of metal.

7. *Form of Casting.*—The shape of the casting is most important, and not as well understood by designers as its importance would indicate. Not only is the piece as a whole affected, but any subdivisions likewise, wherever initial stresses occur.

The joining of members of greater with those of smaller cross-section is an especial cause of weakness. Hard and unyielding cores, or unevenly rammed moulds and sharp internal corners are similarly sources of weakness.

The homogeneity of a casting depends largely upon slow and uniform cooling. A certain element of time seems to be required for the molecules to adjust themselves.

E. D. Estrada, Pittsburgh, Pa.: The discussion in which Mr. Webster has taken the initiative, cannot but prove of great value to the manufacturer and user of cast-iron. Mr. Webster should be congratulated; and in no way can appreciation of his efforts be better demonstrated than by assisting him to carry out the work he has undertaken. I will offer, at this time, a single simple observation on the behavior of cast-iron in passing from the liquid to the solid state.

When a vessel is filled with liquid cast-iron at a temperature somewhat above that at which solidification begins, the volume of the liquid metal diminishes continuously until part of the liquid has reached the temperature at which solidification is possible.

The next phenomenon observable is an increase in the volume of the liquid when it is found that part of it has solidified. This increase of volume is proportional to the rate of solidification. Next to this increase of volume is a corresponding diminution of volume; and we may say that the whole process is accompanied by alternate increase and decrease in the volume of the remaining liquid, which action continues until solidification is complete.

These are the results of my observations on this subject. If similar results have been obtained by others, and the actions described could be verified, it would be a step toward solving a great many of the problems now unsolved, and which cannot be solved unless we search for new physical truths. We may write and speak about shrinkage, chill, blow-holes, segregation, etc.; yet, unless we are familiar with the laws governing the change of state, all our rules and formulas about shrinkage, chill, etc., amount to nothing.

Asa W. Whitney, Philadelphia, Pa.: As a contribution to the investigation suggested by Mr. Webster, the following report of experience in iron-mixing at A. Whitney & Sons' Car-Wheel Works, though substantiating Mr. Webster's conviction with regard to the intimate relation of chemical composition and physical character of cast-iron, may be somewhat disappointing in specific scientific interest. The complexity of the problem, as well as the fact that obvious mercantile reasons are necessarily yet operative, must still excuse the withholding of my general working theory and practice, which is, perhaps, more nearly in the condition of a chemical art than of a science. But a general outline of the results obtained thereby, and some previous history, will be of interest.

A general belief in the value of the chemistry of cast-iron has been entertained at these works since Asa Whitney founded this business in 1847. His interest in the matter led to some study of practical analysis of iron by Mr. John R. Whitney. I find no record of such work, however, in the books of the firm; and any close application of chemistry was considered entirely unpractical. The first analysis (a complete one) was ordered by Mr. George Whitney to throw light on the peculiarities of a certain pig-iron. A test-wheel was cast from this iron May 6, 1875, and the analysis of the same was returned by Humphreys & Wallace, May 28, 1875. Standing alone, however, it was of no value at the time. In 1881, owing to the increasing necessity for general economy and certainty of result, Mr. James S. Whitney urged a more thorough trial of scientific method in the foundry, and the services of Mr. A. E. Outerbridge, Jr., then in the assay department of the United States Mint, were obtained. He had charge of our experimental work and iron-mixing until 1887 inclusive, when his connection with our works ceased.

We are indebted to Dr. Dudley, of the Pennsylvania Railroad, for his suggestions at this time to Mr. Outerbridge, especially in reference to the effect of silicon in car-wheel iron. Until 1888, the only complete analyses of our wheel-metal or pig-iron on our records represent one wheel-mixture and about seven samples of pig-iron. These were all obtained by Mr. Outerbridge from various chemists in 1881.

From 1881 to 1884, the effect of silicon upon the depth of chill of wheel-mixtures and casts of single pig-irons was observed in an experimental way by Mr. Outerbridge, who also carried on various physical experiments of value.

Our laboratory was then a very small affair, no analyses of iron, other than silicon-determinations, being made there until 1884, when I was employed on general analyses, the selection of proper lines of work being left to me.

Apparently, about all that was known in this country with reference to the chemistry of chilling-iron was summed up in Mr. Outerbridge's article, entitled "The Genesis of a Car-Wheel," which appeared in the Philadelphia *Ledger*, August 3, 1883, and in which he refers to the fact that a "variation of less than 1 per cent. of silicon is sufficient to make or mar a car-wheel," and says, concerning chill, that "little is known, even among the most expert iron masters, of the causes that produce it." Mr. Outerbridge's experiments about this period upon chilled ingots of various irons (1 inch square by 4 inches long, and smaller sizes), in reference to their relative transverse strength, fracture, and specific gravity, directed our attention to the wide differences to be found in chilled iron. But our investigation of chemical causes was confined to silicon; and to the total lack of any information with regard to the amount of the other elements present in the cases examined, must be ascribed the impossibility of explaining the characters of these irons, or of practical application of these interesting but partial data to the running of foundry-mixtures by chemistry alone without recourse to physical tests. On one occasion, indeed, the percentage of silicon in a mixture was calculated from the silicon determinations of the irons used, the percentage of silicon in the casting being also determined and compared with the charge; but no further attempt seems to have been made in this line—and wisely, owing to the complications introduced

into such work by the unknown amounts of the other important elements.

Moreover, as has been observed by those who have wrestled with the anomalies of this branch of chemistry, even complete analyses may be misleading. In our case, even as late as 1889, with much more complete analytical work, a close application of chemistry seemed hardly practicable. Only after developing, to a practical point, a general working theory or key by which to use analysis, could the application of chemistry be insisted upon.

Thus, until January, 1892, the irons were tested and mixed entirely on a physical basis, reference being had to name, grade, chill, strength, and general character, as determined by special cupola-melts, and, more recently, by crucible-melts of the irons in question. The suitability, or proper proportion to use in a mixture, was, of course, only shown by an actual trial in the mixture, a risky as well as an expensive procedure at the best.

In February, 1886, appeared Dr. Dudley's paper upon "The Constitution of Cast-Iron,"* in which he gives the analyses of a good and of a poor wheel, respectively.

These analyses did not explain to him or to others the physical differences; and thus, still further doubt was thrown on the value of the ordinary ultimate analysis of cast-iron. As this was exactly the problem upon which I had begun to get some light by the analysis and study of various wheels, soft castings, and pig-irons, I was naturally pleased to find the facts noted by Dr. Dudley fall into line with many similar facts in accord with my incipient working-theory, by which I found all complete iron-analyses becoming intelligible. Even our incomplete analyses, taken in connection with fracture, were to a large degree intelligible. One of the most useful puzzles to work out was found to be the case in which the chill in a casting from two irons poured together was much greater or less than the average chill of these irons taken separately.

Seeing no necessity to study each element singly, I began to apply my theory, *in medias res*, to experimental mixtures, to develop the theory by practice. The success of this line of work at last rendered more economical and certain the manu-

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* *Trans.*, xiv., 795.

facture of special soft mixtures of small contraction and even grain; mixtures whose chilled and gray portions are equally strong; sharply-defined chill, or fine-grained blending chill; close, or open-grained mixtures of good quality, etc.

Predictions of strength of the regular wheel-mixture followed with great success, reference being had to the chemistry only, while the mixture was still worked on a physical basis. In one such case, extending over thirteen days in August, 1890, the predictions of strength of the wheel-mixture for the current day, relatively to the previous day, from a calculation on three elements (silicon, manganese, and phosphorus), and observations of previous days' tests of chill and transverse strength, were right twelve times out of the thirteen, while the expectations, based upon forty-four years' practice, which regulated the mixture by observation of physical data only, failed of realization eight times.

Since January 1, 1892, by the application of my rather crude form of chemical theory, we have been able to safely omit all the physical tests of the component metals of wheel and other mixtures, relying, with far greater success, upon the proper use of the ultimate analyses only of the metal on hand.

On January 11, 1892, in order to further dissipate some doubts of those who could not believe in the new system, I gave the figures for a suitable chemical composition for another trial-melt for January 12th in a separate cupola, requesting a melt to be made of a mixture of the irons apparently most unsuitable, from either a physical or chemical standpoint, that could be figured from their analyses into that chemical composition, with the proviso that if that composition could not be made another would be given. The figures first proposed could not be nearly enough approximated to suit me; so my chemical assistant, Mr. Fisher, who knew nothing about the physical qualities of cast-irons, made some slight changes, making another good composition which I had suggested. None of the irons to be used were selected by me. The opinion of the physical party of 44 years' experience was that the mixture would be entirely unsuitable for wheels, and especially that the chill would be absent or inappreciable. I, however, guaranteed a good mixture with good chill with such close approximation to the appearance of the regular mixture of the same day that

chill-tests from each would be difficult to distinguish, and that the strength of test-wheels and test-pieces would be at least as good as for the regular mixture. The result was successful; the chill-tests of this and of the regular mixtures were closely examined by every one in the office and shop, even those with most experience in iron being unable to detect which were from regular and which from trial-mixtures; and when finally a guess was insisted upon, it was wrong.

The strength of the wheels under Pennsylvania Railroad drop-test was appreciably greater than those of the regular mixture of that day. The chill on wheels was very slightly lower, and the test-bars slightly stronger than for the regular mixture. Chemistry was a success.

As a matter of comparison I may report that special melts of new mixtures in small cupolas had been made previously on the old basis, but with very discouraging results, while the analyses of these casts soon showed the reason for failure.

As an instance of the applicability of our chemical practice I will report a few cases.

A large machine-works sent me at my request half of a long tensile test-bar which they claimed to have cast from a lot of iron-borings melted in a crucible. They had not the analysis at hand, but considered it nothing notable, as "the silicon was normal." This test had probably been cast of about 1¼-inches diameter and turned down to ⅞ inch at the ends.

This iron worked beautifully under the tool and had a peculiar close fracture. Our analysis gave unusual figures.

	Ult. strength, pounds.
Their test of whole piece showed, ...	44,900
Our test near old fracture showed, ...	43,380
Our test about 4 inches from old fracture showed, . .	46,000
Specific gravity, 7.272 at 4° C.	

In a special close-grained cupola mixture which I soon after modelled upon the analysis of the above I obtained 40,070 pounds ultimate strength from a piece turned from half of a test-bar 2 by 2 by 14 inches in size, whose modulus of rupture was 59,925 pounds per square inch and whose resilience was 60. This was slightly hard, but smooth to turn, and took a fine finish. I hope to excel this at my next attempt in that line.

An ultimate strength of 29,000 to 35,000 pounds in metal sufficiently easy to machine does not appear to be difficult to obtain economically from an ordinary cupola. I have several tests between these limits. Our soft iron work will show tests from 17,000 to 31,380 pounds according to requirement.

A notable case is a mixture for bottle-moulds. These are cast-iron moulds used in shaping bottles and other glass-ware. They are used hot, and often under pressure. Our castings of a special mixture to suit the requirement are capable of a life of eleven months, with little sign of distress as yet, while the former moulds, cast by the users themselves, lasted but one or two months. The work put on these moulds is an item of great expense, which makes this longer wear and continued freedom from defects in the polish appreciated, as further orders have proved.

Another instance is that of a special die-casting, wanted at short notice, of strong, not chilled, close, moderately hard grain. In this case, a reference to figures representing the composition of mixtures to be cast that day permitted a calculation showing which and how much of each should be poured together from the different cupolas to give the result. Thus in a few minutes, without the slightest regard to the individual irons, a satisfactory mixture was made of an excellent grain, low shrinkage and ultimate strength of 27,325 pounds. The same method of iron-mixing was used in the case of a gear-wheel, with excellent results.

It will be seen that physical tests are not obsolete, but are applied more rigidly than ever to the product, not only as a guaranty to ourselves and to the public that there is no mistake in the mixtures, but also for study, and to keep track of the relatively small irregularities, some of which would scarcely be considered when mixing iron by physical data, and to learn what kind of irregularities appertain to each kind of mixture. This makes it possible gradually to apply the chemistry more closely and to define the theory more exactly.

As a matter of fact, I usually direct the changes of composition not only without reference to the physical character of the component irons, but even without reference to their analyses. That is, an assistant may figure out from any convenient analyses the chemical change I have directed in a given mixture.

The details of foundry-practice and the regulations of fuel, blast, flux, etc., are of course no less important than heretofore, but rather more so, as they have to be considered in their relation to the compositions charged, and their effects can be made use of to the utmost.

The wasteful and irregular practice, formerly considered necessary at times, here and elsewhere, of cooling the iron in small ladles by a bucket of water, or by a lump of solid iron, or by delay before pouring, is obviated with great advantage in quality, time, labor and material, by making the iron mixture in the cupola of the right character to pour well when hot instead of irregularly when cold.

A point in regard to test-bars may be of interest. I have adopted a bar of hexagon section for transverse tests as more suitable, especially for hard iron, than the square bars of the same sectional area formerly employed, and especially better than the square bar cast with half the area of the ends against an iron yoke, from which the contraction is measured. The iron yoke, of course, chills or mottles the iron at the ends of the bar and causes a strain. The square corners also chill in many cases, and cause further strain, and an irregularity unnatural to the grey iron. In the case of the hexagon bars, on the contrary, the corners are so obtuse that even very high-chill mixtures give an even grain without chill or even "tightness." There is appreciably less surface exposed to varying cooling-influences, when cast. An important point of advantage over a round bar is that there is sufficient bearing on knife edges of the testing-machine to obviate any considerable error in measurement of deflection. The side for a hexagon bar of 4 square inches area is 1.24 inches. By proper gating and casting on end, we get very reliable bars without flaws.

Our daily tests are figured out for modulus of rupture and for resilience. The contraction, transverse strength and resilience of chilled iron bars of wheel-mixture is also daily tested in the same way from the beginning, middle and end of the melt. Also all other mixtures are frequently tested for transverse strength and occasionally for ultimate strength, specific gravity and hardness.

Contraction is measured by a vernier from two small lugs, projecting from an edge of the hexagon. A small yoke, which does not touch the bar, makes a smooth face about $\frac{3}{8}$-inch

square on each lug. The faces are of course 12 inches apart while in contact with the yoke. The volume of these two lugs being less than $\frac{1}{8}$ cubic inch has apparently no ill effect upon the 48 cubic inches of metal between supports for a transverse test. We are making a change in these lugs, however, to improve their accuracy. Tensile tests are usually taken from half of such a bar. In wheel-mixture they would show somewhat higher results if taken from a casting of say $1\frac{1}{2}$ inches diameter.

The hexagon-form is unsuitable for chilled iron. Our chilled test-bars have 4 square inches of sectional area, and are cast 1.5 inches deep by 2.67 inches wide and 14 inches long. They are chilled on the wide sides, having sand on the narrow sides, upon which latter are the lugs by which contraction is measured. A modulus of rupture of 50,000 to 60,000 is not uncommon, with resilience about 30 per cent. of that of the same iron, cast in hexagon bar of 4 square inches area.

As our Mr. C. F. Fisher worked out the mathematical points with reference to the hexagon bar and its relation to the square bar, he has at my request, summarized these points and his experience in testing these bars. I append this matter as a note.*

* *Note on Mathematics of Bars.*—The modulus of rupture of the hexagon of 4 square inches sectional area was calculated from the formula, modulus $= \frac{MC}{I}$ in which (S being the stress) M, or maximum bending moment, $= \frac{\text{stress} \times \text{length}}{4} = \frac{12}{4} S$; $C =$ least distance from outside to center of gravity $= 1.074$ inches; I, or moment of inertia, $=$ the product of the area of section on one side of the center of gravity of the whole into the square of distance from center of gravity of one-half section to center of gravity of the whole section. G, or distance from center of gravity of one-half section to center of gravity of the whole section $= 0.593$ inch. I, for one-half section, $= 2 \times (0.593)^2 = 0.703$, and for whole section $= 0.703 \times 2 = 1.406$. Hence: Modulus $= \frac{S \times 3 \times 1.074}{1.406} = 2.22\,S$.

For square bars 2 by 3 by 12 inches, the modulus of rupture $= 2.25 \times$ stress. The deflection varies as $\frac{I^2}{C}$, and deflection of square bar : deflection of hexagon :: 144 : 134.08 or as 100 : 94.

In practice, however, it is found that the shape of the hexagon bar so affects the grain of the iron that the deflection is actually increased instead of diminished, and that the strength is also increased more than the formulas indicate. Hexagon bars poured from the same mixture at the same time as the regular square bars show an increase of strength of from 10 to 20 per cent. and an increase of resilience in some cases of over 100 per cent., the average being about 50 per cent.; so that it is not usual for a bar of gray wheel-mixture to have a resilience of over 110. It may be well to note that the deflection of the levers of the testing machine has been determined, and is always deducted, to obtain the correct deflection, before calculating the resilience in inch-pounds per pound of metal.

Through the courtesy of Mr. A. E. Outerbridge, Jr., of William Sellers & Co., I have recently had an opportunity to observe temperatures of molten iron by the Mesuré & Nouel's optical pyrometer, described by Mr. F. E. Thompson in the *Iron Age* of February 21, 1895.

The hottest metal from the cupola gave readings, on different days and from different kinds of iron, from 68° to 60°. Wheel-iron pouring from the large ladle gave 63° to 60°; the same into moulds from small ladles, 60° to 55°.

Some practice would doubtless be required before such readings could be guaranteed as closely comparative.

In this connection, I would say that our experience with my chemical system of mixing iron indicates that the most refined methods and processes of all kinds are well worth while, if further knowledge and closer control of the possibilities of cast-iron is to be economically attained. Then I believe such tables as Mr. Webster has worked out for steel can be elaborated with considerable accuracy in this field also. Even now much should be done in the line of more specific tabulation from our own accumulated records, which study would deserve further success.

Henry D. Hibbard, Highbridge, N. J.: The study of the physics, chemistry, and metallurgy of cast-iron, and of the relation of these branches to each other, presents a field of appalling magnitude. The physics, chemistry, and metallurgy of steel, have been much studied for a quarter of a century, yet the subject is so complex, and so much of it is still in the dark, that unexplained discrepancies between these branches are so frequent as to be almost the rule. But if steel is complex, cast-iron is much more so. In the latter we have all the chemical factors which we find in steel, and each exists through a much greater range of percentage.

The following table roughly indicates the usual ranges of the principal chemical ingredients, and of the principal physical properties of cast-iron.

Chemical Constituents.

	Per cent.
Combined carbon,	0. to 5.
Graphitic carbon,	0. " 5.
Silicon,	0.2 " 10.
Mn,	0. " 20.
S,	0. " 1.5
P,	0.01 " 1.5

Physical Properties.

Tensile strength, . . .	1000 to 45,000 pounds per square inch.
Elongation, . . .	0 to 5 per cent.
Compression, . . .	50,000 pounds per square inch upward.

Even the broad divisions of the subject, within some one of which it is necessary to keep to secure commercial results, are very numerous, while the whole number of kinds of cast-iron which may be made, each with an appreciable difference from any other, is practically infinite.

The metallurgy of cast-iron, embracing all the different ways of mixing, melting, treating, etc., with the variations of temperature, moulds and other conditions of manufacture, is thus complicated almost beyond intelligent comprehension. We seem to have first an infinity of causes and then an infinity of effects.

From this it appears that a great number of facts must be collected and classified before much progress can be made in tracing the relationship of cause and effect in this art.

The uses, properties and combinations of properties required of cast-iron are also very widely varied.

The chief property desired or called for by the proposed application of the cast-iron, may be strength, toughness, resistance to abrasion, softness, fluidity when melted, weakness, resistance to corrosion of water or other corrosive agent, soundness, smooth surface, weight, electric permeability, malleability before or after treating, excessive hardness, excessive softness, resistance to compression, or resistance to shock, or a combination of these, or other qualities.

Contributions of facts to the general fund must, to be valuable, give all the circumstances of each case covering chemical and physical properties, as well as conditions of production and use, or they will be incapable of classification, and therefore of little use in solving the question involved.

As a slight contribution from an outsider the writer will venture one or two speculations concerning sulphur in cast-iron. Some things have led him to believe that this element is exceedingly bad for castings in which toughness and strength are desired. Many years ago, some 12-inch square ingot-moulds for casting steel ingots, which gave bad results (some cracking the first time they were used), were found to contain 0.15 per

cent. of sulphur. They were made of Bessemer pig, and in other respects were fairly normal.

This led to the question: Is not the superior quality of charcoal pig-iron due partly to low sulphur, especially in the harder chilling-grades? Charcoal contains no sulphur, and if the ore is free from it, white iron with no sulphur may be made, as is the case with white Swedish pig. Gray pig, with practically no sulphur, is readily made from sulphurous ores and fuels, but in the chilling varieties the case is different, when such materials are used. Again, is not the superior quality of an air-furnace casting due in part to low sulphur? It is well known that sulphur is eliminated from molten pig-iron containing manganese, if it is allowed to stand for some time; and, as almost any mixture of irons would contain some manganese, the conditions in an air-furnace should be most favorable for the elimination of sulphur.

These speculations are capable of proof or disproof, and facts from practical foundry-men bearing on the question would be received with great interest.

NOTE.—This discussion was continued at the February meeting, 1896, in numerous papers and contributions, which will appear in vol. xxvi.

Notes on a Southern Coal-Washing Plant.

Discussion of the Paper of Mr. J. J. Ormsbee (see p. 113).

(Atlanta Meeting, October, 1895.)

WILLIAM B. PHILLIPS, Birmingham, Ala.: The analysis of Pratt coal made by myself, and given by Mr. Ormsbee in his paper (p. 113), is likely to mislead the reader as to the real nature of this coal. It represents the very purest coal that I have been able to get, and does not represent the average composition of the lump-coal as prepared for market. I wish it were true that the coal had no more than 2.30 ash and 0.83 sulphur. The analysis of McCalley is nearer the real figures than any of the others. There is no use in giving an

analysis as representing a coal when it does not do so. It gives a wrong impression of the conditions under which business is conducted. The analysis was made for the purpose of establishing a standard analysis of the very purest coal in the seam, and not for publication as a representative analysis of the coal, lump or otherwise. In studying with Mr. Erskine Ramsey, Chief Engineer of the Tennessee Coal, Iron and Railroad Company, the effects of the Robinson washer on this coal, we endeavored to fix upon a certain analysis and a certain specific gravity as a standard, and after considerable work, we found that with a specific gravity of 1.272, the purest coal had the composition given in Mr. Ormsbee's paper. Since that paper was read, we have continued the work, and can now give some additional data respecting the composition and specific gravity of the coal and the slates. I give the analysis already quoted for comparison.

The following table embodies the results:

Analyses of Coal and Slate from No. 2 Slope, Pratt Mines, Ala.

	Coal.					Slate. Bottom.	Slate. Parting.	Slate. Top.
Specific Gravity..	1.272	1.268	1.307	1.397	1.672	2.303	2.584	2.637
Volatile............	29.80	32.71	28.28	31.08	18.48	11.74	13.10	6.32
Fixed Carbon.....	67.90	63.03	56.89	56.91	44.19	12.84	10.48	2.33
Ash..................	2.30	3.23	13.77	11.55	36.41	74.07	76.42	90.64
Sulphur............	0.83	1.28	1.52	4.65	0.89	0.56	0.61	0.62

The analysis in the first column is the one quoted by Mr. Ormsbee. The last analysis under the heading *Coal*, is of "Bone Coal." The analyses are on the dry basis.

In no case has the specific gravity of the slates been as low as 1.80. It generally runs from 2.30 to 2.60.

In discussing the effectiveness of washing a given coal, it must be borne in mind that there is a great difference between essential ash and sulphur and accidental ash and sulphur. Essential ash is the ash which must be regarded as a constituent part of the coal, not to be removed by any known method of treatment without destroying the integrity of the coal itself. Accidental ash is the surplus over and above this amount, and may be removed. The extent to which the accidental ash may

be removed depends upon the nature of the coal and the means employed.

Some writers have used the term "organic ash" to denote the ash which is a constant part of the coal; but this is clearly a misnomer, for there can be no such thing as organic ash; it is a contradiction in terms. All coal-washing schemes are based upon the hope of freeing the coal from accidental ash, for there is a certain amount of ash which is, so to speak, a vital constituent of the coal, and not to be removed from it. (The same is true of sulphur, except that there may be some sulphur organically combined.) Just what this amount may be is to be determined not only for each coal, but for each grade of coal offered to the market, with or without washing. Where only the slack-coal is washed, and the question is as to the amount of ash and of sulphur that can be profitably removed from it, of course we have to do merely with this kind of coal, and need not concern ourselves with the composition of the lump- or nut-coal. Given a certain slack-coal, the question is, to what extent the ash or sulphur may be profitably diminished, not removed entirely, for there must be left enough ash in the coal to give strength to the coke. If it were possible to remove the sulphur entirely, the coke would be of better quality, and would command better prices; but this is not possible in the case of sulphur any more than in the case of ash; for even the purest coal contains some sulphur, and it appears to be essential to the very coal itself.

In the case under discussion, the slack-coal, before washing, contains, on the average, 9.98 per cent. ash and 1.48 per cent. sulphur. How much must be regarded as essential ash and essential sulphur? What amount above this can be regarded as still within the limit of economical work? We have seen that the essential ash in the purest coal obtained from the seam is 2.30 per cent. But it would evidently be requiring the washer to perform the impossible, to ask that it should take slack-coal and bring the ash in it to the limit of the ash in the purest coal. The percentage of ash in the purest coal has, therefore, but little to do with the problem.

If we take the lowest amount of ash found in the slack as 6.08 per cent., and allow 1.67 tons of washed coal to the ton of coke, the percentage of ash in the coke will be close to 10

per cent. by calculation. From a large number of analyses of coke made from washed slack, I have found that the average ash is 10.13 per cent., which is very close to the calculated ash, and shows that the probable limit of the ash in the washed coal is about that of the ash in the best slack. In other words, it would appear that when the slack has been washed until its ash is 6 per cent., we have reached the limit of profitable work. I do not say that this is absolutely true; but it seems to me to indicate that the essential ash in the slack is close to 6 per cent. From samples representing several thousand tons of coke, and taken over several months by the same person in the same manner, it has been found that the average ash is 10.13 per cent. In like manner, it has been found that it takes 1.67 tons of coal for 1 ton of coke, and that the average ash in the coal is 5.78 per cent.

As regards the sulphur, the question is somewhat different. The essential sulphur in the slack cannot be given with the same degree of accuracy as the ash. I am disposed to take it at 1.25 per cent., although the average is 1.48 per cent. in the slack. The sulphur seems to follow the fine coal—a fact of very general application; for, with the exception of some Westphalian coals, I do not recall any coal in which the sulphur follows the coarser particles. The Robinson-Ramsay washer seems to be specially adapted to the economical treatment of slack carrying heavy slate and not overburdened with sulphur, although some excellent results have been reached with slack carrying more than 2 per cent. of sulphur and even 2.50 per cent. As such, this washer has proved of the greatest value in the Birmingham district, and has before it a future of unusual promise.

The Lixiviation of Silver-Ores by the Russell Process at Aspen, Colorado.

Discussion of the Paper of Mr. W. S. Morse (see p. 137).

(Florida Meeting, March, 1895.)

C. A. Stetefeldt, Oakland, Cal.: It has always been assumed by the writer, and also by others, that the silver volatilized by roasting in a Stetefeldt furnace was a minimum as compared with roasting in other furnaces. Now, Mr. Morse records at

Aspen a loss of 9 and even over 10 per cent. Without doubting in the least the accuracy of Mr. Morse's statistics, I wish to say that it is not safe to draw general conclusions from a limited experience.

Unfortunately, I have no array of statistical figures at my disposal to combat Mr. Morse's statement, if he means to apply it to the Stetefeldt furnace at large, but must confine myself to general arguments.

In the first place, where ores are mined and reduced by the same company, accurate statistics in regard to the dry weights of the raw and roasted ore are not kept. This, for example, is the case at the Ontario and Marsac mills, Park City, Utah. In *Trans.*, xiv., 341, I have recorded experiments establishing the comparative loss of silver in roasting Ontario ore in the Howell and Stetefeldt furnaces. In one of these experiments the loss of silver was 13.5 per cent. greater in the Howell than in the Stetefeldt furnace. Now, if the Stetefeldt furnace had lost 10 per cent. silver, the loss in the Howell furnace would have been 23.5 per cent., which is not probable. In following the argument in my paper quoted above, it will be seen that the loss in roasting can be determined approximately by an indirect method. In the chloridizing-roasting of base ores a considerable change in weight takes place, *i.e.*, the roasted ore, minus soluble salts, weighs considerably less than the raw ore before roasting. In comparing the value of the raw ore with the calculated value of the roasted ore minus soluble salts, the latter value should be greater in proportion to the change in weight, provided silver has not been lost. This change in weight can be determined by roasting average samples of ore in the muffle. From numerous determinations made in this way, it appeared that the loss of silver by roasting Ontario ore in the Stetefeldt furnace would not exceed from 2 to 3 per cent.

Several mills using the Stetefeldt furnace have done custom-work, namely, the Reno mill and the Manhattan mill in Nevada, and the Lexington mill in Montana. Although I have no statistics from these mills at my disposal to prove my case, the following general argument will, nevertheless, have some weight.

The Reno mill did custom-work exclusively, the Nevada Land and Mining Company having no mines of its own, and

the business was very profitable while the supply of ore lasted. No complaint ever reached me as to extravagant loss of silver in roasting. Mr. Ottokar Hofmann was employed there for some time, and, since he never was an enthusiastic admirer of the Stetefeldt furnace, he would surely have published facts detrimental to its reputation if results had warranted him in so doing. The Manhattan mill, Austin, Nevada, did custom-work most of the time, and sometimes exclusively. After the introduction of the Stetefeldt furnace, this mill not only found a great saving in the cost of roasting, but also a higher percentage of extraction as compared with roasting in reverberatory furnaces. Having bought the exclusive right to use the Stetefeldt furnace in the Reese River district, they reduced their working-charges, and soon monopolized the buying of ores. Later on, an opposition mill was again started with a Howell furnace. It had to shut down on account of inferior extraction, *i.e.*, as shown by Ontario experiments, great loss of silver in roasting. I have no Manhattan mill statistics to show; but would the Manhattan company have paid a royalty on every ton of ore put through the Stetefeldt furnace, for fifteen years, if it had lost 10 per cent. of silver in roasting?

The Lexington mill at Butte, Montana, has often been running exclusively on purchased ores, and, as I understand, with profit. Mr. Rueger, the general manager of the company, a graduate of Freiberg, and an able metallurgist (but not a member of the Institute), has never complained to me about losing money in consequence of having Stetefeldt furnaces at his mill. At the time I made the comparative test between roasting in Howell and Stetefeldt furnaces at the Ontario mill, I applied to Mr. Rueger for statistics on the loss of silver in roasting at his mill; but he did not see fit to grant my request.

In conclusion, I offer some explanations of the large loss of silver by roasting in the Stetefeldt furnace, at Aspen, as reported by Mr. Morse.

In the first place, the quantity of ore roasted per day (from 90 to 92 tons) was unusually large; and considering that 8 per cent. of sulphur in sulphurets had to be oxidized, it follows that strong draft had to be used. No long flue connects the last dust-chamber with the chimney at Aspen, as is the case at the Ontario mill. Under the circumstances, it is my opinion

that a more extensive system of dust-chambers would have collected a large portion of the dust and fumes passing out of the chimney.

In the second place, the character of the Aspen ore is highly abnormal, on account of the high percentage of lime and magnesia it contains. Investigations at Aspen have shown, according to Mr. Morse, that the percentage of lime and magnesia in the roasted ore is largely reduced, and it must be assumed that these metals are volatilized as chlorides, their oxides not being volatile at high heat. It seems to me highly probable that these chlorides carried silver with them and thus caused, in part, the abnormal loss at Aspen.

Mr. Morse: Replying to Mr. Stetefeldt's remarks on the loss of silver in roasting Aspen ores with a Stetefeldt furnace, as recorded in my paper, I would say that it was not my intention to attack the Stetefeldt furnace, but simply to report the actual results obtained at Aspen.

It is unfortunate that so little reliable information has been published on this very important operation in the treatment of silver-ores either for amalgamation or for lixiviation; and it was this lack of published data that prompted the writer to give the Aspen results, in the hope that a discussion would follow which would bring forth the experience of others. Mr. Stetefeldt's "general argument," however, does not fill the bill. His experiments, as recorded in *Trans.*, xiv., 341, showing the comparative losses in roasting in Howell and Stetefeldt furnaces, offer a very round-about way of attacking a very simple problem; and he does not arrive at the all-important fact of what the loss actually was with either furnace. The fact that Mr. Hofmann did not publish facts detrimental to the Stetefeldt furnace, does not show that the loss of silver in roasting in a Stetefeldt furnace is not large.

I sincerely hope that others having reliable statistics on the subject will publish them.

The results at Aspen, as published, are made up from records very carefully kept, and the calculations are based on the average of thousands of samples and assays; so that an error in a few samples or assays would not affect the general result, and it will require more than "general argument" to convince me that I am in error.

The results of a series of experiments conducted by Mr. E. B. Kirby in the chloridizing-roasting of Aspen ores, in a reverberatory furnace, will be of interest in connection with this discussion.

The ores were practically the same as those treated at Aspen, and the average of lime (CaO) was 12.3 per cent., the highest being 25.5 per cent. Eighteen lots of about 1 ton each were roasted in a reverberatory furnace, with salt, for a sufficient time to give fully as high "chlorination-tests" as the ore roasted in a Stetefeldt furnace. Each lot was weighed, sampled and assayed before and after roasting, and the average loss in roasting, including the dust-loss, was found to be 6.64 per cent.

Mr. Stetefeldt's opinion that part of the silver-loss is caused by the volatilization of lime and magnesia may be correct, although this did not seem to be the case in Mr. Kirby's experiments in roasting in a reverberatory furnace, where the silver-loss on the ores carrying 25.5 per cent. of lime was only 5.7 per cent.

It is undoubtedly true, however, that lime and magnesia are volatilized in the chloridizing-roasting of ores. A series of experiments made by Mr. J. Dawson Hawkins at Aspen, in which four lots of ore were roasted with about 12 per cent. of salt, each lot was weighed and sampled, and lime- and magnesia-determinations made before and after roasting, showed a loss of 22.6 per cent. of the lime and 48.3 per cent. of the magnesia during the operation of roasting.

The Tin-Deposits of Durango.

Discussion of the Paper of Mr. W. R. Ingalls (see p. 146).

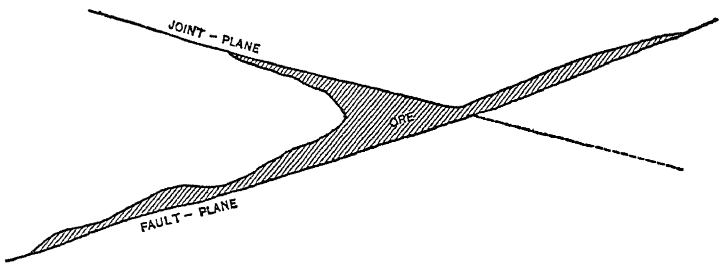
(Atlanta Meeting, October, 1895.)

C. W. Kempton, Oro Blanco, Ariz.: In his interesting paper, Mr. Ingalls refers to reports of tin-deposits near Sain Alto, Zacatecas, Mexico. During 1891 I made some personal examinations of these deposits. They are, for the most part, located from 10 to 30 miles west of the town, in the Sierra del Fraile.

The formation is the same rhyolite as that described by Mr. Ingalls, carrying chalcedony and related forms of silica.

The ore occurs in a multitude of small veins, from a mere seam up to a yard in width, and rarely traceable as ore-bearing for more than a few rods in length. These seams or veins stand nearly vertical, and usually strike northerly and southerly. They appear to occupy fault-planes in the rhyolite.

In one instance only, I noted the occurrence of ore at the intersection of a joint and a fault-plane. The accompanying sketch presents a horizontal section, on the scale of 8 feet to



Occurrence of Tin-Ore near Sain Alto, Mexico.

the inch. At this place, about one ton of cassiterite was taken out. No attempt was made to follow the body in depth. Indeed, the deepest working I saw in the district had not reached 50 feet.

The placer-deposits of cassiterite are undoubtedly supplied from the numerous small veins in the rhyolite, and do not necessarily indicate that any of these veins are sufficiently persistent and extensive to be mined with profit. I doubt whether any workable deposits will be found in the rhyolite, unless, possibly, at its contact with some other formation.

The Assay of Silver Sulphides.

Discussion of the Paper of Mr. H. van F. Furman (see p. 245).

(Atlanta Meeting, October, 1895.)

Albert Arents, Alameda, Cal.: From Mr. Furman's description of his crucible-assays I infer that he regards iron nails as a necessary or advisable adjunct. Against such a notion I must beg leave to protest. Nails should not be used in

this method of assaying for silver. They not only do no good but may be a cause of loss, especially in operating upon material so rich as silver sulphides. It is difficult to conceive for what purpose iron nails would be introduced into a crucible in such a case except that they may take up sulphur, forming therewith a matte. But any matte so formed will contain some silver, which is thus withheld from the regulus and lost to the assay. Even if no matte is formed, the nails are detrimental, because they must be withdrawn, coated with argentiferous lead from the molten contents of the crucible before pouring. This constitutes another loss which might be avoided.

On the other hand, if no metallic iron is employed in the assay, but a sufficient amount of litharge is added to oxidize completely the sulphur in the charge, leaving at the same time some excess of oxide of lead, both the sources of loss above indicated are avoided. The result of such a crucible-assay, properly conducted in other respects, would be worthy of comparison with that of scorification. The question of the relative accuracy of the two methods is interesting and deserves further investigation, in which, however, both should be handled with equal care and favor, so that each may accomplish the best results of which it is capable.

H. Van F. Furman, Denver, Colo: As regards the use of nails in the assay of silver sulphides, I agree with Mr. Arents, that they are unnecessary. The amount of sulphides present is so small that the litharge used will serve all purposes. I cannot agree with Mr. Arents, however, that nails should never be used in the crucible-assay of silver-ores. In the case of ores rich in sulphur, arsenic, or antimony, I regard metallic iron as a most useful flux. With such ores, it becomes necessary to add either metallic iron or some oxidizing agent more powerful than litharge, such as niter. Where litharge alone is used to effect oxidation, a large amount must be used, and, should the ores be rich in sulphides, such a large button will be obtained that repeated scorifications will be necessary before it can be cupelled. Should niter be used, it becomes necessary to determine the reducing power of the ore. This requires a preliminary assay; and the assayer at a modern reduction-works has no time for preliminary assays. In our Colorado practice, niter

is additionally objectionable as, in order to avoid the bubbling over of the charge, large crucibles are necessary. Large crucibles cannot be used where the fusions are performed in the muffle, as is the Colorado practice. When the ores are rich in sulphides, the use of metallic iron necessarily involves the production of matte. However, this should not result in loss of precious metal, as the matte should be saved and scorified, together with the lead button, prior to cupellation. That the use of nails results in loss from their being coated with argentiferous lead when withdrawn from the crucible, is only true in the most careless work. No careful assayer will withdraw the nails without first carefully washing them off in the slag so as to remove any adhering particles of lead. When the work was carefully performed, I have never known the use of nails to result in loss. It is for the above reasons that nails are almost universally used in Colorado when the ores are rich in sulphides. A nail was added to the charge in the case of the assays under discussion, in order to make the method conform to the usual practice.

As I stated in my paper, these crucible assays were not made because this was considered the proper method for such material as silver sulphides, but in order to draw attention to the relative accuracy of the crucible- and scorification-methods. Such being the purpose, the assays were made in the same manner as would be pursued in careful commercial work. I am inclined to believe that such material may be preferably assayed by the combination-method, as outlined in my paper.

Assays of Copper and Copper Matte.

Discussion of the Results presented at the Florida Meeting, March, 1895 (see p. 250).

(Atlanta Meeting, October, 1895.)

W. M. Courtis, Detroit, Mich.: Except for free-gold ores, I have always believed in scorification. I think the crucible-assay gives silver low. We found this out at the Wyandotte works in assaying Silver Islet ore. The mine-managers made us, and also the check-assayers, run one of the duplicates by

crucible. As we took the mean of 12 assays for ore running from 700 to 10,000 ounces of silver per ton, they soon found that the crucible-assay reduced the average result. So it was left out, and scorification was employed for all the assays.

CABELL WHITEHEAD, U. S. Mint, Washington, D. C.: After studying, in connection with my own experiments, the results reported by the various assayers co-operating in Dr. Ledoux's plan, I have no doubt that the results obtained by the combination-method are too low, and while this loss can be greatly reduced by using dilute acid, I have been, up to the present time, unable to devise any method of preventing it entirely. The results show that scorification gives about one-tenth of an ounce per ton more gold than the combination-method. I consider this difference greater than it should be if dilute acid be used, but even a few hundredths of an ounce is not to be disregarded, when the total amount present is usually less than half an ounce.

In the case of silver the results are not so conclusively in favor of scorification. While the figures obtained are fully up to, and in some cases beyond, those given by the combination-method, the additional work of assaying slags and cupels, as well as the doubt concerning the purity of the buttons, would suggest the desirability of a less laborious and more accurate method. While a second cupellation with a "proof" greatly reduces the chance of error from impure buttons, I am not sure that it entirely removes that chance, as the fire-assay of silver is far from satisfactory.

I am inclined to believe that the combination-method as modified by "J" leaves little to be desired. The advantages are, first, the use of bromide, instead of chloride, as a precipitant for silver; second, there need be no corrections for loss of silver in slags and cupels (the amount of silver cupelled being not more than 5 or 10 mg., these losses may be disregarded without introducing an appreciable error); third, a larger sample can be taken than is possible with scorification.

In a large copper-works, recently visited, I observed a great improvement in the method of parting small amounts of gold. The buttons are placed in small test-tubes, and 20 c.c. of dilute nitric (1:7) was poured in each. The tubes were set in a beaker of cold water, which was then put over the lamp and

brought to boiling. With acid of the strength specified the gold did not break up, but retained the shape of the button, remaining as a black ball. The weak acid was now poured off and 20 c.c. of 32° B. acid was added, and the water in the beaker was boiled for 10 minutes longer, when the gold was washed and taken out in glazed porcelain crucibles. When this method is used it makes little difference how small a proportion of gold the buttons contain.

H. VAN F. FURMAN, Denver, Colo.: The results as presented by the different assayers taking part in the investigation are of great interest to those engaged in gold- and silver-assaying, and should result in benefit, even if they do not lead to the adoption of some standard method for the assay of the material under consideration.

The wide divergence in the results as reported shows that an investigation of this nature was timely. Through analysis and discussion of the results it appears to the writer that much may be gained, and it is hoped that the discussion may be full.

That some of the reported results are unquestionably too high and others too low is apparent. Without going into a complete analysis of these, I venture to point out some possible sources of error:

Some of the assayers have stated that the test-lead and litharge used were free from silver. The writer does not believe that any commercial test-lead or litharge in the market is entirely free from silver. Gold is rarely present; but silver is invariably present, at least in the experience of the writer. The lead which I am using at present shows 0.3 mg. of silver in 100 grammes. This would be equal to about 1.33 oz. of silver per ton, reported in the ordinary scorification-assay of an ore or matte, when $\frac{1}{10}$ A.T. of the sample was taken. This lead is sold by one of our standard dealers and is made by one of our oldest and most reliable refining-establishments. I am informed that the lead was treated with zinc seven times in order to extract the last traces of silver. Every one who is familiar with zinc-desilverization knows that it is impossible to remove the last traces of silver, even should the zinking be continued almost indefinitely. In order to determine the silver in the test-lead it is not sufficient to take one A.T. and cupel it. More frequently than otherwise the result would be no button of

silver. From 100 to 300 grammes of the lead should be scorified until a lead-button of not over 3 grammes is obtained. This button is then to be cupelled quite cold for silver. The same is true of the litharge; but the ordinary litharge usually contains more silver than the test-lead, and its silver-contents are often unevenly distributed.

Another cause of too high results, where the proper precautions are not taken, is the imperfect elimination of the base metals on the cupel. This is especially the case with material high in copper, as is instanced by some of the reported results in this case, where the silver buttons were shown to contain over 3 per cent. of copper. The only safe rule is to eliminate nearly all the copper from the lead-buttons prior to cupellation, or to run a synthetic proof and make corresponding deductions, as was done in some of the instances reported. Should the lead-buttons contain considerable copper, the latter method is at best only an approximation, as it is impossible to estimate closely the amount of copper present. In all cases the cupel should be pushed back into a hot part of the muffle, just before the "blick."

Another source of error, probably more common than is generally supposed, is due to the cupel used. If the cupel is made of too coarse bone-ash, or has not been compressed sufficiently, it is liable to be porous, and the cupel-absorption will be quite heavy. I notice in one case reported that 15 ounces of silver (to the ton of matte) was recovered from the cupel; certainly an excessive amount. The writer uses XX bone ash, and gives the cupels such pressure that when thoroughly dry they may be dropped from a height of about 3 feet without danger of breaking.

Another important point is to heat the cupel to the temperature of the muffle, and when it has been thus heated, open the door for a few minutes before dropping in the lead-button. This serves to destroy any organic matter which the cupel may contain, and which might cause the button to "spit."

Another important point, and one to which the writer has more than once traced differences in assays, is the inaccuracy of the milligramme-weights ordinarily sold by the dealers and manufacturers. It is extremely difficult to obtain sets of these weights which are concordant among themselves, and are exact

subdivisions of a gramme. Two instances of this have recently come to my notice. A set of gold-assay weights from 1000 (= 0.5 gramme) to 1 (= 0.5 milligramme), consisting of 20 pieces, was ordered from a prominent manufacturer. On receipt of the weights, I found that the 1000 weight was not only not exactly 0.5 gramme, but that there was quite a wide difference among the individual weights. Some of these weights were too heavy by quite 0.3 milligramme. The weights were returned to the maker with the request that they be made correct. When I received them again from him, I found them more nearly correct, but some were still so far from accuracy that I had to readjust them. Of over 100 riders recently examined, and which were marked 1.2 milligrammes, every one was found to be too light. The weights as purchased should never be used until they have been subjected to a thorough test and found to be correct.

In addition to the set of my own results contained in the report presented to the Institute by the Secretary, the following determinations, made on the same material after sending in my statement of assays and methods, may be of interest:

	Silver. Oz. per ton.	Silver. Oz. per ton, average.	Gold. Oz. per ton.	Gold. Oz. per ton, average.	Copper. Per cent.	Copper. Per cent. average.
Matte	133.6 132.5	133.05	2.25 2.30	2.275	54.28 54.13	54.20
Copper	153.3 153.2	153.25	0.28 0.26	0.27	97.65 97.23 98.08 97.65	97.65

The matte was first ground to pass a 120-mesh sieve; the material as received being entirely too coarse for accurate analysis or assaying.

The method employed for the determination of the copper in the matte was the cyanide titration-method, essentially as described by "K," aluminum being used as a precipitant. The copper in the drillings was determined as follows: 1 gramme of drillings was dissolved in a flask with 10 c.c. of nitric acid (32° B.), and the contents were heated until the red fumes had been expelled. The solution was then transferred to a 500 c.c.

graduated flask and diluted with distilled water to exactly 500 c.c., and thoroughly mixed. 4 portions of exactly 100 c.c. (0.2 gramme) each were drawn off with a pipette, run into a 200 c.c. flask and 15 c.c. of strong ammonia was added to each portion. The solution was then titrated with a standard solution of potassium cyanide, each c.c. of which was equivalent to 0.005 gramme of copper. With each set of assays a blank-assay should be run, using pure copper foil and the same amount of nitric acid, ammonia and water as in the regular assay. This is important, as the standard solution is liable to change in strength. The proof-assay also presents the advantage that it is run side by side with the regular assays, and the same conditions are introduced in all.

A comparison of these results with those obtained by the electrolytic method would lead to the conclusion that the titration method is quite as accurate as the latter, if not more so. Thus, in the case of the copper-borings, the difference between the highest and lowest of the four volumetric determinations is much less than in the case of the average electrolytic determinations, whilst the average of the four (97.65 per cent.) is very close to the average of the electrolytic results (97.67 per cent.). In the case of the matte, while the volumetric determinations are somewhat lower than the average of the electrolytic determinations, the writer believes that they more truly represent the amount of copper present. This matte contains some antimony and zinc, and doubtless other elements, which are liable to be precipitated with the electrolytic copper, and thus give too high results. I quite agree with assayer "N," who remarks: "For such material (the matte) I do not believe electrolysis to be as good a method for determining copper as some others."

The assays for gold and silver were made as follows, the method being the same for both matte and borings: One A.T. of material is introduced into a No. 5 beaker and 100 c.c. of distilled water is added. The mass is stirred with a glass rod and 50 c.c. of nitric acid (sp. gr. 1.42) is added. As soon as the action of the acid has ceased 50 c.c. more of nitric acid is added and the solution is gradually heated until the red fumes have been expelled. The solution is now diluted with distilled water to 500 c.c., and allowed to stand several hours. After thorough settling, the solution is filtered off through a rather

heavy 4½-inch filter-paper; the water used in transferring the precipitate to the filter being generally sufficient to wash out the silver- and copper-salts.

To the filtrate, which should be perfectly clear, normal salt-solution (1 c.c. representing 10 milligrammes of silver) is added in slight excess. (To the matte 14 c.c. and to the borings 18 c.c. was added.) The solution is stirred, 10 c.c. of a saturated solution of lead acetate is added, and the solution is stirred again. Now, 2 c.c. of sulphuric acid (1 part concentrated acid and 1 part water) is added, and the solution is stirred. After allowing to stand several hours, the precipitate is filtered off and thoroughly washed with distilled water. The filtrate should be perfectly clear. The filters are removed from the funnels, wrapped around the precipitates and placed in 2½-inch scorifiers, the paper containing the first residue being placed in the same scorifier with its corresponding silver-lead precipitate. The filters and precipitates are now dried by placing the scorifiers on a hot plate, and when dry the paper is burned by placing the scorifiers in front of the muffle. The scorifiers are finally pushed back into the muffle to destroy all the carbon and sulphur. To each scorifier is now added 5 grammes of litharge, 15 of test-lead and one of borax glass. The charges are then scorified and poured, the resulting lead buttons weighing about 6 grammes. The buttons are cupelled so as to show feather-litharge and are pushed back into a hot part of the muffle just before the "blick." After weighing, the buttons are parted for gold by flattening them, introducing each button into a small porcelain crucible and adding nitric acid (15° B.). After all action of the acid has apparently ceased (it should be at boiling temperature) it is poured off and fresh acid of 32° B. is added. The gold is boiled in this acid 4 minutes, when the acid is poured off and the gold is washed twice with distilled water. After drying, the gold is annealed and weighed.

The writer quite agrees with assayer "D" that the combination silver-assay is more accurate, and it certainly requires no more time or work than the scorification-method, especially when, in the practice of the latter, the slags and cupels are saved and assayed for the absorbed silver.

J. N. Walker, Everett, Wash.: After reading the report of

the various assays of samples of copper and copper matte presented at the Florida meeting, I wish to offer some further statements bearing upon the use of a proof-assay. After determining by the scorification-method upon the sample of matte received from the Secretary, the commercial results, such as would be reported in the course of ordinary business, I extracted the silver from the scorifier-slags of four sets of assays. No. 1 (slags from one scorification) gave me 0.3 ounce silver per ton; No. 2 (two scorifications), 0.6 ounce; No. 3 (three scorifications), 1 ounce; No. 4 (two scorifications, very cold), 0.8 ounce. This seemed to indicate clearly that with this grade of matte, the silver-loss in successive scorifications was practically uniform.

The loss by absorption in the cupel appeared to me most serious of all; and I was led to believe also that this loss was the greater, the greater the proportion of copper in the buttons to be cupelled. Thus, the results from 3 buttons were as follows:

No.	No. of Scorifications.	Copper in buttons. Ounces per ton.	Silver in Cupel. Ounces per ton.
1	1	3.03	3.7
2	2	2.95	3.53
3	3	2.23	3.20

These cupellations were made at the same time and at the same heat, but the buttons were the product of one, two and three scorifications respectively; and the reduction in copper by successive scorifications is attended with a corresponding reduction in cupel-absorption of silver. The elimination of copper by scorification, however, is not proportioned to the number of scorifications made. In my experience the second scorification is the most effective in this respect, though in the instance here given, the third has produced a greater diminution.

But the chief conclusion to which I have been brought is that the heat of cupellation has more to do with the loss of silver in the cupel than anything else. I give herewith some results of experiment in this direction, which, however, I regret to say were not checked by duplicates:

No.	Scorifications.	Cupellation.	Silver. Ounces per ton.
1	Hot.	Hot.	118.6
2	"	Cold.	128.2
3	Cold.	"	130.6
4	"	Hot.	120.4

The indication is clear that excessive heat in cupellation causes excessive loss of silver by cupel-absorption.

After weighing my beads, and also the silver obtained from the cupels and from the scorifier-slag, I decided to cupel them again with a small amount of lead, and to run at the same time for comparison a pure silver-button of about the same weight. The result was as follows:

A set of three buttons, resulting from two scorifications each, the first with 40 grammes of lead, and the second with 20 grammes of lead plus the button, weighed 39.5 milligrammes. The proof-button weighed 39.12 milligrammes. After cupellation of both at medium heat (the main point being to get rid of lead and copper), the weight of the assay-button was 37.6, and that of the proof 38.1 milligrammes. The normal cupel-absorption of silver, under the circumstances, was, therefore, as 39.12 : 38.1 or 1.027 : 1. Multiplying 37.6, the new weight of the assay-button after cupellation with 2 grammes of lead, by 1.027, we have 38.615 as the probable weight of the three buttons before this cupellation, if they had been pure silver. The difference between this and their actual weight, 39.5, is 0.885, a loss which cannot be fairly ascribed either to cupel-absorption or to volatilization. Dividing it by 3, we have 0.295 milligrammes (or 2.95 ounces per ton for each button of original assay), as the loss of weight, due, in my judgment, to copper in the silver-buttons; and I therefore determined the actual amount of silver in the assay as follows:

		Ounces per ton.
In original average weight of button,		131.6
In scorifier-slag,		0.6
Silver absorbed by cupel,		3.53
		135.73
Less: Copper in button,	2.95	
Gold "	2.34	
		5.29
Silver contents,		130.44

In ordinary commercial practice, the copper in the button and the silver in slag and cupel would have been neglected, and the total weight of the silver-button, less the gold determined by parting, would have been reported as the result in silver.

ALBERT R. LEDOUX, New York City: The variations among different assayers in the report of these comparative determin-

ations is what I expected, and emphasizes, more than any words of mine could do, the importance of our "getting together." In my opinion the differences exhibited are due to the choice of methods, and do not reflect upon the skill and care of the participants in this joint inquiry or "symposium." I trust that the discussion will be continued with the fulness and freedom which the subject deserves.

There is one question only here and there touched upon in the reports of the several assayers which is bound to come to the front in business practice very soon, namely, Shall allowance for cupel-absorption, volatilization or loss in slag be made in commercial reports of silver assays?

In copper-assaying the returns aim to represent the total copper present; and the allowance for subsequent loss in smelting is fixed by an arbitrary and uniform agreement or usage between shipper and buyer or smelter.

In gold-assays, likewise, the reported result is understood to show the total contents of gold in the sample; and the buyer is accustomed to receive one-eighth or one-quarter of an ounce per ton free, to compensate for loss in smelting, and also to pay for the remainder of the gold at less than its full value as refined metal (say 90 or 95 per cent. of this value, according to the agreement he may make).

But in the assay of copper or copper matte containing silver, it is probable that the reported value in silver always comes short by an ounce or two per ton at least, of the actual total contents in that metal, by reason of the general practice on the part of commercial assayers of ignoring the losses to which I have referred. The question is, whether this practice shall be continued, or whether the assay-report shall be corrected by an allowance for this loss.

It seems to me that the matter is one to be settled primarily by agreement between buyers and sellers, rather than by the assayers who stand between them. But it is unquestionably important that whatever is determined upon should be adopted by all those interested in public sampling-works or in the business of commercial assaying. Otherwise, there would be interminable disputes arising, not out of lack of skill or care in either party, but out of a difference in the rules of practice.

Pending the settlement of this question, it is highly advan-

tageous that the nature and extent of such assay-loses should be investigated, and that the most accurate and convenient method of allowing for them should be determined.

I feel greatly encouraged by the interest shown in this subject, and trust that the result will substantially benefit both the metal trade and the guild of assayers.

Folds and Faults in Pennsylvania Anthracite-Beds.

Postscript to the Paper of Mr. B. S. Lyman (see p. 327).

(Atlanta Meeting, October, 1895.)

In reply to inquiries and comments which have reached me since the publication of this paper, and in explanation of some seeming discrepancies between my statements in the text concerning the amount of displacement of the faults, on one hand, and certain figures in the plates on the other hand, I give here, with some additional remarks, my measurements of the displacements in the different sections, which it originally seemed unnecessary to print:

Plate.	Section.	Feet.	Plate.	Section.	Feet.
II............	F	25	VII........	1	35
II............	20	50	VII........	1′	75
III............	20	90±	VIII........	7	165 ?
IV............	28	95	IX........	36	100
IV............	10	135 ?	X........	28	90
IV............	12	130	XI........	35	370 ??
IV............	11	240	XI........	39	35
V............	19	700 ??	XIII........	12	110
V............	19	50	XIII........	25A	380 ??
V............	B	105	XIV........	25	380 ??
V............	C	35	XIV........	13	?
VI............	18	35	XV........	40	60
VI............	18	25	XV........	40′	(700)*
VI............	B	40	XXVI........	14	20
VI............	A	10	XXVIII........	8	?
VII............	18A	20	XXX........	10	115 ?
VII............	4′	10			

* Overturn.

By displacement, I mean neither the throw, *i.e.*, the vertical distance apart of the two edges of a faulted bed (what Mr.

Bailey Willis calls "the vertical throw"), nor the heave (which he calls "the horizontal throw"), *i.e.*, the horizontal distance along the fault from one edge or end of the faulted bed to the other; but the distance along the dip of the fault between the two said edges or ends.

As to the displacements in Plate XI., Section 35, and Plate XIII., Section 25 A, marked 370 and 380 feet, respectively, it seems quite doubtful whether they are really so great. The one on Plate XV., Section 40, which is marked 700 feet, seems to be more properly an overturn (and, in fact, is so drawn), with perhaps a fault of very doubtful extent.

Of the large displacements, there remains only to be considered the one on Plate V., Section 19, marked 700 feet.

In that case, it appears, on the examination of the mapping of the mine-sheet (Northern Coal-Field, Part I., sheet vi.), that, owing to the irregular crumpling of the rock-beds, the section-line at that particular place is, for some distance, nearly parallel with the strike. A section at right angles to the local strike would show a much smaller displacement. The map, however, does not seem to give all the information necessary for a complete section. Perhaps some of the lines were omitted, on account of the danger of too great confusion in representing workings on the same bed at two different levels. Or, I may fail to understand the lines perfectly because the two sets are not distinguished by dotting or otherwise. Apparently, the displacement would be, at most, less than 250 feet.

The concluding statement of my paper is based on the 26 measurements of displacement which were not affected with very great doubt, excluding those which are doubly marked (??) as doubtful in the foregoing table.

It will be noted that normal faults appear but seldom on the sections of the Pennsylvania Geological Survey. One reason may be, that such faults would be more or less nearly parallel to the section-lines. Moreover, their usual small extent would probably render them inconspicuous in small-scale sections.

Recent Phosphorus-Determinations in Steel.

Discussion of the Paper of Mr. G. E. Thackray (see p. 370).

(Atlanta Meeting, October, 1895.)

T. M. DROWN, South Bethlehem, Pa.: Mr. Thackray's paper shows in a striking way the high degree of rapidity and accuracy exhibited by the chemists of our steel-works in ordinary everyday practice. That determinations made in an hour should present only slight differences in the third place of decimals is certainly matter for surprise and congratulation. This remarkable result has been due, perhaps, in part, to a remarkable cause. The managers of works, ignorant of all the difficulties of the laboratory, have demanded "the impossible" of their chemists—and the chemists have done it!

ANDREW A. BLAIR, Philadelphia, Pa.: Mr. Thackray's paper is extremely interesting, especially in its bearing on the question of a standard method for the rapid determination of phosphorus in the laboratories of steel-works. Before discussing the results in general, I desire to explain the results obtained by me under method A'. This is the method which I have used in the work of the sub-committee on Methods of the International Steel Standards Committee. It may be described briefly as follows:

Solution in nitric acid; oxidation of carbonaceous matter by permanganate; precipitation of phosphoric acid by molybdate solution; reduction of molybdic acid in the reductor, and titration by permanganate. This method was used apparently by only three chemists, Dr. Dudley, Mr. Crowell, and myself. In determining phosphorus by this method, among the most essential points to be considered are the ratio of iron to molybdic acid, or the reducing action of zinc on molybdic acid as compared to its reducing action on iron, and the ratio of molybdic acid to phosphorus in the ammonium phospho-molybdate obtained by precipitation from the solution of the steel.

In regard to the first point, the usually accepted ratio is 1 to 0.90756, or, as it is roughly stated, 0.9076, which is based on the assumed reduction of the molybdic acid to $Mo_{12}O_{19}$. This

is probably true as regards the reduction by zinc in Emmerton's method, but is not true when the reductor is used. In a paper prepared by Mr. J. Edward Whitfield and myself for the American Chemical Society, the details of our investigation of this reaction are given, with our reasons for taking the reduction to be to $Mo_{24}O_{37}$, and the ratio 1 to 0.88163. The results obtained by me and by Mr. Crowell under the method A′ were calculated by the old factor. Corrected they would be:

Blair's results:

Sample No.	Phosphorus. Per cent.
19,915,	0.0495
19,533,	0.0816

Crowell's results:

Sample No.	Phosphorus. Per cent.
19,915,	0.0515
19,533,	0.0835

In regard to the second point, the ratio of molybdic acid to phosphorus in ammonium phospho-molybdate is 1 to 0.01794. Dr. Dudley, however, in his method (while he does not mention it in this paper), has assumed this ratio to be 1 to 0.0190. In my opinion this position is untenable. His results recalculated to the ratios 0.88163 and 0.01794, are:

Sample No.	Phosphorus. Per cent.
19,915,	0.0495
19,533,	0.083

The twenty-seven methods described in this paper fall under two heads:

1. The acetate method (A).
2. The molybdate method.

The latter is again divided into five distinct methods:

1. Titrating the reduced molybdic acid after passing the solution through the reductor (B).
2. Titrating the reduced molybdic acid after boiling with metallic zinc and filtering (C).
3. Weighing the precipitated ammonium phospho-molybdate (D).
4. Redissolving the precipitated ammonium phospho-molybdate in ammonia and precipitating the phosphoric acid as am-

monium magnesium phosphate, with final weighing as magnesium pyrophosphate (E).

5. Determining the amount of ammonium phospho-molybdate by ascertaining the quantity of standard alkali required to neutralize it (F).

An analysis of the results obtained by the different methods is given in the annexed table. No. 1 is sample 19,915. No. 2 is sample 19,533.

	Method A.		Method B.		Method C.	
	Sample No. 1.	Sample No. 2.	Sample No. 1.	Sample No. 2.	Sample No. 1.	Sample No. 2.
Number of results......	3	3	3	3	2	2
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Highest....................	0.049	0.081	0.0515	0.0835	0.049	0.083
Lowest.....................	0.046	0.078	0.0495	0.0816	0.045	0.080
Average....................	0.048	0.080	0.050	0.083	0.047	0.082

	Method D.		Method E.		Method F.	
	Sample No. 1.	Sample No. 2.	Sample No. 1.	Sample No. 2.	Sample No. 1.	Sample No. 2.
Number of results......	6	6	11	11	4	4
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Highest....................	0.052	0.086	0.055	0.089	0.053	0.086
Lowest.....................	0.049	0.083	0.046	0.076	0.046	0.078
Average....................	0.050	0.084	0.049	0.083	0.049	0.081

The greatest variation is found under method E, the molybdate-magnesia method, giving a difference of 0.009 for sample 19,915, and 0.013 for sample 19,513. This is to be expected from the latitude in treatment under this method.

The interesting points, however, are that the volumetric results compare so favorably with the gravimetric, and that the averages under each method show such slight variation, the greatest variation being 0.003 in sample 19,915, and 0.004 in sample 19,513.

The averages of the entire number of determinations (with the changes referred to above in method B) are 0.049 for sample 19,915, and 0.082 for sample 19,533.

It is much more satisfactory to have results obtained by differ-

ent methods confirm each other, than to have a number of agreeing results by the same method; therefore, the close agreement between the results obtained by the acetate method and those obtained by the molybdate method is very gratifying. The average by the acetate method is 0.049 and 0.081 (excluding the low results obtained by Dr. Rothberg, as they are considered only experiments by Mr. Thackray). The average of all the averages of the molybdate methods is 0.049 and 0.083. On the whole, these results show a decided advance in accuracy over any previous series, and give strong hope that the adoption of a single method will eliminate all errors, except those due to carelessness.

Southern Magnetites.

Discussion of the Paper of Mr. H. S. Chase (see p. 551).

(Atlanta Meeting, October, 1895.)

E. C. Pechin, Buchanan, Va.: I am sorry to see the table appended to Mr. Chase's excellent paper. In the discussion at the same meeting, on "Notes on a Southern Coal-Washing Plant," Prof. Phillips very properly says: "There is no use in giving an analysis as representing a coal when it does not do so. It gives a wrong impression of the conditions under which business is conducted." The same may be said about ores. I have no personal knowledge of the ores in many of the localities given in Mr. Chase's table; but if these are to be judged by what is positively known of the Cranberry ores, the analyses are grossly misleading. The furnace-shipments from Cranberry have rarely reached 44 per cent. of iron, and any such percentage as 58 or 66 or 68 can only have come from hand-picked specimens. Indeed, with any such percentages, Mr. Chase's paper would have been superfluous, because concentration would be unnecessary. Besides, the paper itself (page 553) gives the fact that the Cranberry average is 42 to 43 per cent. Why should a table be appended, contradicting this statement of the text?

Let us always have the exact conditions under which business may be conducted. "Boom" and business analyses are widely different.

MR. CHASE: Mr. Pechin's criticism is just. The analyses given in my table were made from hand-picked specimens, and run from 10 to 20 per cent. higher in iron than furnace-shipments would show. Presented without further explanation, as "a table of representative analyses," it might be misunderstood, and I regret that it is too late to amend my final paragraph by adding an explicit statement of the real significance of the table, namely, its "representative" character as showing the proportions of sulphur and phosphorus in selected rich specimens of certain Southern magnetites, and thus indicating what might fairly be expected, as to these two elements, from equally rich concentrates of such ores. From this standpoint, I believe the figures given in the table to be suggestive and valuable.

As Mr. Pechin points out, I have plainly said in my paper that the Cranberry ore averages only 42 to 43 per cent. of iron. Moreover, I have spoken in conclusion of "the low percentage of iron and the large percentage of silica" in southern magnetites; and, since the subject of the paper was the preparation, from such material, of Bessemer concentrates, I think the inference is obvious that my table giving analyses of rich specimens was not intended to represent the average mine-products which need to be concentrated, but their probable nature after concentration.

While I frankly accept the criticism, therefore, I think it fairly applies, not to the introduction of the table, but to the lack of the explanation which I left the reader to infer.

Present Condition of Gold-Mining in the Southern Appalachian States.

Discussion of the Paper of Messrs. Nitze and Wilkens (see p. 661).

(Atlanta Meeting, October, 1895.)

ADOLPH THIES, Haile Gold-Mine, S. C.: I have little to add to what my friends Messrs. Nitze and Wilkens have said on this subject. What I have done in mining during the last forty-one years, partly in this country and partly in South Africa (though not, as to Africa, in the diamond- or the gold-fields,

but in the copper-belt of Namagua land), has been recorded so fully that I need say nothing more about it here.

With regard to the development of the chlorination-process which I undertook years ago, and the result of which has come to be associated with my name, I beg to say that I have accomplished but little, yet that little is based upon the facts of actual practice, and has put this process in such a form that its operation requires no skilled labor. An ordinary negro can be taught in a week to perform the manipulations as well as any expert.

I have brought here from the Haile mine some specimens of a rock which we have always looked upon, and which has been generally spoken of for years past, as talcose slate. While Messrs. Nitze and Wilkens were at the mine, I was driving on the 270-foot level a diagonal drift, which exposed this apparently talcose slate, as judged from its greasy "feel." But analysis showed it to contain only 0.22 per cent. of magnesia. At first, I could hardly accept this result; but when a second analysis confirmed the first, and when the diabase itself was shown to contain 7 per cent. of magnesia, I was compelled to believe that what we had classed as magnesian slates were in fact clay-slates.

In these slates occur layers of pyrites, from a few inches to several feet in thickness, and sometimes alternating with the rock for a considerable distance. But in general the pyrites occur as an impregnation of the rock. Assays have shown that where it is interlaminated with the slates, the rock is unprofitable. Moreover, when the drillings do not feel sharp and gritty, the ore is generally of very low grade. In other words, the more siliceous ores are the ones for us to work. As a further guide in this respect we have adopted daily "panning," with occasional fire-assays. The latter alone would not be trustworthy if made simply on mine-samples. Of course, the mill-tailings and concentrates, as well as the chlorination-tailings, are regularly sampled and assayed.

By reason of the great width and average low grade of the ore-bodies, we have adopted, in mining, the pillar-system, without the use of any timber. We leave pillars from 15 to 20 feet thick between the stopes, extend the stopes for 30 to 35 feet in length, and mine the material between hanging- and

foot-walls as far as safety will permit. True walls have not been reached on either side. Whenever we find the rock losing its siliceous character and assuming a slaty habitus, we know that we are in unprofitable ground. And when we fail to find any free gold by panning, we know that the sulphurets also will be too poor to work profitably. Yet we are obliged to extract and work large quantities of such material, which are so mixed with the siliceous ores as to make a separation impracticable.

All the stopes are connected directly with the surface, so that they may be hereafter filled with surface-material, and the pillars now left standing may be completely extracted.

The present depth of the main shaft is 280 feet, of which 10 feet is a sump. At each level we enlarge the shaft by driving on each side, so as to have a water-storage for about forty-eight hours, in case of any accident to the pump. When sinking is resumed, the water is kept out of the shaft proper by means of batteries, and the two sides are connected by a pipe, so that a pump on one side easily takes care of all the water down to the last level. In this way the sinking itself can be done with the use of a bucket only.

The total underground excavations at the Haile mine to-day, including drifts, cross-cuts and winzes, are equivalent to between one-half and three-quarters of a mile of linear work of ordinary dimensions. The reserves at the present time would supply a 60-stamp mill for at least three years.

In the 60-stamp back-to-back mill, built by the Mecklenburg Iron Works, of Charlotte, N. C., we have 32 square feet of amalgamating-surface to each battery of 5 stamps. Tests have been made with different meshes, and it has been found that wire screens of 50 meshes to the inch are the best for this particular material. The inside battery-plates are taken out every 24 hours, for the reason, as I suppose, that the slime made from the sulphurets covers the surface of these plates, so as to reduce their effectiveness in amalgamation. At all events, we have found that when plates are left in the battery for a week we get less aggregate amalgam from them than when they are taken out every day.

The pulp, as it leaves the batteries, is made to strike first on an impact-plate, not for the sake of an extra saving of gold on

that plate, but rather for the purpose of retarding the velocity of the discharge. If the pulp passed directly from the battery to the apron-plates its velocity would be so great that little gold would be saved on the upper plates. After it leaves the stationary 8-foot plates it drops upon three 12-inch plates, so arranged that the second lies at an angle of about 30 degrees below the plane of the first, and so on. From the third plate the pulp passes into the launders. These are provided with a series of riffles, which have been found very effective in catching loose quicksilver and amalgam.

We use in the battery as little water as practicable, just enough to get the sand out. I prefer to thin the pulp after it leaves the battery. The launders carrying the riffles are about 80 feet long, have ½-inch fall to the foot, and empty into square boxes, whence the pulp is distributed by small launders, 2½ by 2 inches in size, to the Embrey concentrators, of which we have 20.

Our experience with concentrators leads us to prefer the end-shake to the side-shake. The latter keeps the fine slime too much agitated. Our concentrates contain up to 90 per cent. of pure pyrites. Our loss is 15 to 20 per cent. We cannot catch it all.

The concentrates are hauled to the furnace for roasting. For this purpose, I must say that, in my opinion, the old reverberatory is the most reliable apparatus. I use the double-hearth furnace, 40 feet long, and also the horizontal revolving hearth. The product is the same in quantity, each furnace giving 2 tons of roasted ore in 24 hours. As 3 men are required to do this with the pan and four with the reverberatory, the economy is a little in favor of the former. The roasting is done by black men, and I may here say that during 35 years in the South I have found black labor the best, and so consider it to-day. I have now about 75 black men employed, and pay them "white" wages. Even in tending machinery I can trust them as fully as white men. As a hammer-man in the mine the negro cannot be surpassed by any other nationality. But he does not like to work with an English miner, because he has to strike through the whole shift. The English miner wants to hold the drill all the time!

The average amount of concentrates roasted is 200 tons per

month, yielding about 150 tons of roasted product. The men test the product for themselves with the aid of an old oyster-can and a test-rod. If I find that a charge is not properly roasted they have to pay for the wood wasted and re-roast the charge without extra pay. On the other hand, they are paid a dollar for every ton roasted beyond the minimum required, and they manage to get out five or six tons a month extra from each furnace. The roasting must leave not more than ½ of one per cent. of sulphur in the product. It would cost too much to make a dead roast more perfect than that.

The concentrates carry $25 to $30 gold per ton. I have had concentrates here that ran as high as $60 per ton. The average value of the crude ore as mined is $4 per ton. I have mined ore as poor as $3, and even $2.75. But no profit can be got from such material except by strict economy and steady mining on a large scale. This is the cardinal question with our southern gold-ores: quantity. When the supply of ore is abundant and regular even a low-grade may be profitably treated. But a precarious or limited supply, coupled with low grade, is a fatal combination.

R. W. Raymond, New York City: The distinction between the schistosity, or secondary foliation, of the rocks of the regions described in this admirable paper and their stratification or original bedding, is one of the results of careful and detailed field-work on the part of trained observers, which must modify to a large extent the views of earlier authorities and throw light upon many of the problems they encountered. In the case of roofing-slates, for instance, the fact that the foliation, due to pressure and furnishing the predominant cleavage, does not follow the planes of stratification, has long been recognized. But in the crystalline schists it has been unquestionably the general practice to identify the foliation with the bedding. I fancy that, until within a few years, the observations of geologists as to the dip and strike of gneiss-beds have been well-nigh universally based upon the position of the schistose layers. Moreover, the schistosity has been in many instances assumed as a proof of sedimentary origin. This, of course, it cannot be, if it does not represent the bedding-planes. For, if a given schistose structure can be produced in a sedimentary rock without reference to the planes of sedimentation, it might be

produced, under conceivable conditions, in a rock which had no such planes of original deposition.

The bearing of this distinction upon ore-deposits and their origin is not less important. Many of the deposits in the South have been considered as lying parallel with the stratification of the country-rock, and therefore to have been probably laid down with it as intercalated members of the geological series, or segregated from it by metamorphic agencies. But if such deposits are parallel only to a schistosity which is not the stratification, then our view of their genesis may have to be wholly revised.

Mr. Nitze (communication to the Secretary): In connection with specimens of the so-called "talcose slate" exhibited by Mr. Thies, and with the footnote on page 667, in which we speak of the erroneous character of the name, I would say that I have submitted to Dr. Charles Baskerville, the assistant chemist of the North Carolina Geological Survey, for analysis, several specimens from the Haile mine. The results are shown in the following table, which includes also, for purposes of comparison, an analysis of the diabase at the Haile mine.

	I.	II.	III.	IV.
SiO_2,	61.02	51.31	44.61	45.75
Al_2O_3,	25.54	17.50	31.57	13.97
FeO,	4.46	2.04	3.55	16.23
MnO,	......	......	0.16	0.52
CaO,	0.60	0.25	0.20	8.90
MgO,	0.14	0.23	0.22	7.15
Na_2O,	2.19	2.80	6.96	6.02
K_2O,	1.81	1.90	6.97	1.06
Cr_2O_3,	......	......	none.	trace.
FeS_2,	......	19.63	......	......
H_2O,	4.20	4.20	5.80	0.30
	99.96	99.86	100.04	99.90

I. Greasy mineral from specimen of soft greenish schist, Haile mine.

II. The remainder, after most of the greasy mineral had been removed.

III. Soft, greasy, drab-colored schist from the Haile mine.

IV. Diabase from the Haile mine.

It will be seen that the diabase contains much more magnesia than the rock supposed to be "talcose;" and that the latter is undoubtedly sericitic or hydro-muscovite schist.

Wm. B. Phillips, Birmingham, Ala.: In discussing this very timely and valuable paper, one is at a loss to account for the discrepancy between the richness of the mines and the smallness of the return from them. According to the figures given on page 787, it has cost $535,000 to produce $318,000 worth of gold, and the capital invested was $5,900,000. That is, for each dollar obtained there was expended $1.68. At this rate it would not take long to bankrupt the entire country. Considering the Southern gold-fields as a whole, there is no reason to doubt the accuracy of the returns. But is not this way of looking at the matter somewhat unfair? Can we take the entire territory under discussion and charge up against it all the foolish and unwarranted expenditures that have been made in mining and hunting for gold? Judged in this manner, what district would show a profit? Excluding some of the richest gold-fields in this and other countries, we may well doubt whether any of the rest would make a much better showing.

While it cannot be denied that in the majority of cases investments in Southern gold-mines have not been profitable, I think it is time to inquire whether this has been exclusively the fault of the properties, or whether some of the blame may not attach to the management. As I remarked in a paper read before the Institute at its meeting in Buffalo in 1888,* the South has been the foraging-ground for almost every kind of "process-man" known to the public, and is to-day not altogether free from this class of individuals. As a rule the mines have not been profitable; but as a rule they have not been managed by men who knew their business, and who were honestly interested in trying to save the gold. Stock-jobbing operations have been far more in evidence than economically administered establishments; and this has gone on until solicitations to engage in the mining of gold in the South are not entirely free from the danger of a commission *de lunatico inquirendo*.

Now, what are the facts in the case, brought out in the plainest manner by this notable contribution of Messrs. Nitze and Wilkens?

First, that not all the mines are good mines; second, that there are a few really valuable mines; third, that the best mines

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* *Trans.*, xvii., 313.

demand the closest attention to details and the most unflagging devotion to the pay-roll.

The Haile mines, in South Carolina, are often cited as a shining instance of the possibility of making money in gold-mining. While I have only the warmest friendship for our friend and fellow-member, Adolph Thies, the manager of the Haile, and cannot say too much in praise of his administration of this property, still I do not think the instance fairly taken. The Haile stands almost alone in the size of its ore-bearing bodies, and the consequent ease with which they are mined. The ore is not of a high grade, seldom reaching a total value in the ground of $5 a ton, and is sulphuretted, only about one-third of the gold being in free-milling condition.

The problems that Captain Thies has faced and solved are not such as can reasonably be expected elsewhere, so far as I am aware, unless it be at the Brewer, or at the Russell mine. If I were bent upon studying the conditions most likely to be encountered in the South, I would select the old Phœnix mine, near Concord, N. C., where the seam is not so large as at the Haile, but the average conditions are conserved to a better degree. After an inspection of the Haile, one is apt to scorn a little 3- or 4-foot seam, and to doubt the advisability of entering upon a project that offers so little. But the Phœnix was operated for several years very successfully by Captain Thies himself, and it was there that the only reliable method of treating low-grade sulphurets was devised and put into actual use on a large scale. The Phœnix is the father of the Haile; and while the son has greatly outgrown the father, it may be that he had more to eat and less to do.

If would-be investors in Southern gold-mines are looking for another Haile mine, with its enormous ore-bearing bodies of free-milling and sulphuret-ore, I fear they will be disappointed. Such deposits are few and far between, and it is perhaps just as well that they are, for, were they more frequent, the smaller seams would receive scant attention. And yet it is to these latter that the gold-mining industry in the South must apply itself.

I have never been much of a believer in the placer-mines of the South. With a few rare exceptions, this system of mining is not applicable to our deposits, if for no other reason, because

riparian rights of long standing would certainly interfere with the working of gravel on the large scale. If one is so fortunate as to be within easy reach of streams of considerable size, such as the Broad, the Chattahoochee, the Tallapoosa, or the Coosa, there might be an opportunity for the successful carrying-on of placer-mining. But the smaller streams are already pretty well settled, and farming interests would be as apt to interpose obstacles here as they actually have done in parts of California.

It seems to me that there are three principal considerations that must affect profitable gold-mining in the South, and any intending investor would do well to bear them in mind. First, the seams are of moderate size; second, they are of moderate richness; third, each locality must stand for itself as respects the method of treatment.

I am firmly persuaded that nearly all of the failures in this part of the country have been due to the neglect of one, if not all, of these considerations. Until they are borne in mind, and reports and plans are based on a most careful investigation of the local conditions, there can be no notable improvement in the state of affairs. Each property must be examined in and for itself, not only with respect to the size and extent of the deposit, but also and especially with respect to the nature of the ore and the condition in which the gold is found. After passing the water-level (and the distance from the surface to this point varies in almost every locality) we have to do with a mixture of free-milling ore and sulphurets of moderate richness, seldom reaching the total gold-contents of half an ounce per ton. Copper, arsenic and antimony, while occasionally present, do not materially affect the question; and the method to be adopted for the working of the average Southern gold-mine has to be based upon the total amount of gold present, the degree of sulphurization and the relation subsisting between the gold in the free-milling and in the sulphuretted condition. It is not meant that the gold itself occurs as a sulphuret, but that a very considerable amount is present in the pyrite and cannot be secured until this compound is broken up. The degree of comminution of the gold is also a very important point, and one to which it is difficult to devote too much attention.

It may be said that almost any sane man going into the gold-

mining business would investigate these points in the most careful and thorough manner before erecting expensive machinery; but this is the very thing the average investor does *not* do, and the consequence is that the entire Southern gold-fields are left to enjoy the unsavory reputation gained by such foolishness.

I can but repeat what I have said on more than one occasion, viz., that in spite of the numerous failures and in spite of the opinion so widely held as to the unprofitableness of Southern gold-mines, I firmly believe that there are good properties in the South which await but the skill and the honesty of ordinary business management to prove their value.

In concluding, I must take occasion to thank Messrs. Nitze and Wilkens for their valuable contribution to our knowledge of the Southern gold-fields. It is the most notable paper on this subject that has been presented to the Institute in many years, and cannot fail to receive the compliments it so well deserves.

MESSRS. NITZE and WILKENS (communication to the Secretary): With regard to Dr. Phillips's opinion that the Haile mine is not a fair representative instance of profitable gold-mining in the South, because it stands, as he thinks, almost alone in the size of its ore-bearing bodies, and that the Phœnix, a quartz-fissure, 3 to 4 feet in width, would be a juster type of average Southern conditions, we beg to say, that on the assumption that the latter proposition is true, the Haile mine may still be cited with propriety as an instructive example of a dividend-paying enterprise in this field. And we may point out that we have also cited, as another example, the Franklin mine, which represents structurally a different type, namely, that of the Phœnix.

We have certainly not held up the Haile type of large bodies of low-grade ore as representing average conditions. It is simply a well-developed and well-managed mine of its own class. On the other hand, we are not prepared to accept the Phœnix as an average type. In our opinion, a careful examination of the field would perhaps discover as many deposits of the one class as of the other. We heartily agree with the declaration of Dr. Phillips, that each deposit must be examined and judged for itself, with regard to "the size and extent" thereof, as well as the other particulars he mentions.

Dr. Raymond: At the request of Mr. Thies I exhibit samples of a curious film of metallic gold occurring in the chlorination-works at the Haile mine. These works contain 13 precipitating-tanks, in which the solution of gold-chloride is treated with ferrous sulphate to precipitate metallic gold. During last summer Mr. Thies observed that in two of these tanks, but not in any of the others, small scales collected upon the surface of the liquid after 24 hours of the precipitating-process. These scales had the luster of gold and were found upon re-solution and precipitation to be gold. As the contents and treatment of all the tanks were exactly the same, this exceptional phenomenon was attributed by Mr. Thies to the only circumstance which seemed to distinguish the two tanks to which it was confined, namely, that they were so placed as to receive the rays of the summer sun through a skylight, while the remaining tanks were in shadow.

In response to his request for an explanation, which he declares himself unable to give, I venture to offer the following as a suggestion.

The floating of gold-scales on water is a well-known phenomenon, and is generally attributed to the adherence of air to them, diminishing the specific gravity of the total mass. The rising of gold to the surface of a liquid in which it has been set free naturally suggests a similar cause, namely, the adherence of air or gas to the particles of gold. If there is, therefore, any gas likely to be generated in small amount by the action of the sun's rays in the precipitating-tanks at the works in question, the probable cause of the rising of a small amount ot gold to the surface is indicated. No such gas can be presumed to be liberated from ferrous sulphide, gold chloride and pure water; but I am informed that the water used at the Haile works is by no means pure, but contains a great deal of organic matter. This might, I think, liberate gas under the circumstances described. Such a liberation need not be so abundant as to give visible bubbles (though, perhaps, such bubbles might be seen if closely looked for). A very little of it would suffice to float a very little gold.

I take the liberty, reverting here to the contents of the paper before us, of calling attention to the use of the hydraulic elevator at the Chestatee mine, Lumpkin county, Ga., mentioned

on pages 739 and 740. What we saw, during the excursions connected with the Florida meeting, last spring, of the hydraulic exploitation of phosphate-deposits without the aid of natural "head," either for the production of water-pressure or for the escape of tailings, would naturally suggest the possibility of such a treatment of gold-bearing sands and gravels. Indeed, the idea itself is not new. Every engineer knows that he can get the equivalent of hydrostatic "head" by burning fuel to make steam for pressure-pumps, and that he can, in the same way, replace or supplement the action of gravity in removing tailings by lifting them. It is a question of cost, and the cost depends, not only upon local conditions, but also upon the suitable design and arrangement of apparatus. I wish we could have, for our *Transactions*, a more complete description of the method and means employed by Mr. Crandall at the Chestatee mine.

W. R. CRANDALL, Dahlonega, Ga.: In response to Dr. Raymond's wish, I will undertake to present, at the next meeting of the Institute, a paper containing the information he desires.

The Effect of Washing With Water Upon the Silver Chloride in Roasted Ore.

Discussion of the Paper of Mr. Willard S. Morse (see p. 587).

(Atlanta Meeting, October, 1895.)

L. D. GODSHALL, Everett, Wash. (communication to the Secretary): In Mr. Morse's interesting paper several statements occur which seem to require correction and comment. This is the more appropriate, since Mr. Morse has mentioned me by name, and has cited my "Review of the Russell Process,"* so that silence on my part might be taken as indicating an assent which I cannot give.

On the first page of his paper Mr. Morse says that the "going-back" of chlorination was "the only difficulty encountered at Aspen," and adds that he has "never seen any explanation" of the cause. The plain inference is that he now fills this complete

* *Proc. Col. Sci. Soc.*, vol. iv., p. 306. Read May 1, 1893.

lack of an explanation by the hypothesis which he advances. I propose first to correct the statement of fact, and then to examine Mr. Morse's hypothesis and proofs.

An explanation on this point is advanced in Mr. Stetefeldt's book on *The Lixiviation of Silver Ores by Hyposulphite Solutions*, to the effect that steam, generated by wetting the hot ore, might decompose the silver chloride, and leave as a result metallic silver, insoluble in hyposulphite of soda, but slowly soluble in Russell's solution.

Another explanation, suggested by Mr. Croasdale, the chemist of the company, and mentioned by me in my "Review," etc., cited by Mr. Morse, was, that the trouble was caused by the presence of ferrous sulphate. This was for a long time, I believe, the view of Mr. Morse himself.

Still another explanation was advanced by me in the paper referred to, and this is in fact mentioned by Mr. Morse, though he does me injustice by omitting from his summary of it the word "partly." My theory was that the variations noted are partly (not wholly, as Mr. Morse would lead the reader to infer) occasioned by the fact that there is present in the roasted ore a reducing-agent, in the form of sulphurous acid, either free or combined, and that this agent, acting on the silver chloride, reduces it to metallic silver, insoluble in sodium hyposulphite, but slowly soluble in Russell's extra-solution.

It must be concluded that Mr. Morse meant to say that he had "seen no explanation" which was in his opinion satisfactory. That, of course, he had a right to say.

Mr. Morse further asserts that, while superintendent of the works, I "could devise no method to overcome the difficulty" which he declares to have been the only one encountered. On the contrary, I did devise and earnestly recommend a remedy, and I think I shall be able to show from Mr. Morse's own words that the failure to adopt this suggested remedy was the cause of the complete disaster which subsequently overtook the company after scarcely a year and a half of operation.

I must observe in passing, however, that the efficacy of a remedy is not necessarily dependent upon precise accuracy of theory as to the cause of trouble. For instance, the effect of reducing-agents might be successfully counteracted even though those agents were not all correctly recognized.

Mr. Morse declares that, in his opinion, "the cause of the 'going-back' of chlorination is principally the presence in the roasted ore of sulphides of the base metals which have escaped oxidation in roasting; and in the case of the Aspen ores this result was due to the zinc sulphide, which, during the rapid roasting in the Stetefeldt furnace, did not have sufficient time to decompose." In other words, he lays the whole trouble to the Stetefeldt furnace; and it follows that if some other furnace had been used, which could have been so controlled as to continue the roasting until the zinc-blende had been decomposed, the remedy would have been at hand. This suggestion is advanced as novel, two years after the closing of the works. But while they were still in operation, I made in my "Review of the Russell Process" the following statement of the case:

"About the time the Aspen works were first started, the writer was informed by one of the leachers who had previously been connected with the leaching-department of the Blue Bird mill in Montana, that much trouble had been experienced there on account of the water rendering a considerable portion of the silver chloride insoluble in the sodium hyposulphite solution. This water is used to dissolve and wash out the soluble chlorides of the base metals, and is called the first wash-water. The information was received with a smile of incredulity, but in a very short time it changed from a matter of amusement to a stern reality, as it was discovered that this same trouble existed to an alarming extent with the Aspen ores, as then mixed and roasted. A sample of the roasted ore charged into the vat would show a high extraction in the assay-office with 'hypo'- or 'ordinary' solution, as it is generally termed. At times the extraction by the 'ordinary' would be almost as high as with the 'extra'- or Russell solution. But after washing the charge thoroughly with water and removing all soluble salts, and then again taking and leaching a sample, a vast difference would be found between the assay-office 'ordinary'- and 'extra'-extraction. The 'extra'-extraction on the washed ore, as compared with the tests made on the dry ore, differed somewhat; but the 'ordinary'-extraction on the washed chloridized ore would sometimes fall as low as the 'ordinary'-extraction on the raw ore. Innumerable experiments were made to overcome this difficulty, and various theories were suggested as the cause of it. Ferrous sulphate was at one time supposed to be responsible for the trouble, but repeated tests failed to show the presence of either soluble ferrous or ferric compounds. The theory proposed and still adhered to by the writer is that the variations noted are partly occasioned by the fact that there is present in the roasted ore a reducing-agent in the form of sulphurous acid, either free or combined. This reducing-agent, acting on the silver chloride, reduces it to metallic silver, insoluble in sodium hyposulphite, but slowly soluble in Russell's 'extra'-solution. This theory is strengthened by the fact that when a maximum amount of air was admitted into the furnace, thus oxidizing more of the sulphur in the furnace and allowing less sulphurous acid to be formed in the lower heat on the cooling-floor, the difference between the 'ordi-

nary'-extraction on the dry and washed ore-samples was not so great as before. The difference in the extraction on the two samples by the 'extra'-solution was also less.

"As the pyrites used at this time contained quite a large percentage of zinc and lead in the form of blende and galena, it was thought possible that all the trouble might be traced to these impurities. This supposition seemed to be confirmed when, after the introduction of a purer pyrite, results were obtained which proved much more satisfactory; however, while the zinc and lead were responsible for the trouble to a great extent, they were so only in an indirect manner. It is a well-known fact that it is far more difficult to roast zinc-blende and galena than iron pyrites; consequently, when a large percentage of these minerals is present in the pyrites, more sulphur remains unoxidized in the ore after it has passed through the furnace, and the subsequent chloridizing action on the cooling-floor, under a limited supply of air, necessarily produces a larger quantity of sulphurous acid.

"*The irregularities mentioned are also indirectly due to the conditions that obtain in the Stetefeldt furnace. If it were possible to obtain a perfect chloridizing atmosphere, or oxidize all of the available sulphur in the ore while in the furnace, the difficulty with the washed ore would undoubtedly be overcome, as there would then be no reducing agent present to affect the silver chloride. Such a condition it is impossible to obtain in the Stetefeldt furnace when roasting ores carrying as much lime as the average ores of Aspen do.** The question then arises, whether such ores can be roasted better with any other furnace? Metallurgically, the answer is 'Yes.' The proof of the above assertion with regard to Aspen ores has been demonstrated by the writer, by roasting in a reverberatory furnace ten lots of ore containing 25 per cent. of CaO (MgO not determined, but probably amounting to 10 to 12 per cent.) with less than 2 per cent. of sulphur, and using practically the same amount of salt as in the Stetefeldt furnace. The chloridization was all that could be desired, and no trouble was experienced with the washed ore."

It is evident that Mr. Morse agrees with my earlier statement in the following particulars: (1) That the trouble was caused by an incomplete roasting; (2) that this incomplete roasting was due to the action of the Stetefeldt furnace; (3) that it is impossible, with the Stetefeldt furnace, to roast properly such ores as were treated at Aspen; and, I presume Mr. Morse will agree (4) that one remedy is to roast such ores in a furnace in which the sulphur can be thoroughly oxidized.

On the other hand, I do not admit that undecomposed zinc-sulphide is the sole source of trouble; that sulphurous acid has nothing to do with it; or that Mr. Morse is the discoverer either of the cause or the remedy.

It will be observed, that, in my statement, I mentioned zinc-blende (together with galena) as responsible for the trouble to a

* These *italics* are not in the original paper.

great extent. Mr. Morse, in support of his view that zinc-blende alone was the cause, says:

"No difficulty was experienced in roasting in the Stetefeldt furnace charges of ore containing from 16 to 18 per cent. of iron pyrites (8 to 9 per cent. of sulphur), and completely oxidizing the sulphur."

This is not strictly accurate, even when the roasting in the Stetefeldt furnace is considered as including the subsequent heap-roasting of from three to four days on the cooling-floor; since, after such subsequent roasting, upon digging into the hot ore, the smell of SO_2 was almost always noticeable, and frequently so strong as to annoy the workmen.

Mr. Morse says:

"The fact that the final apparent extraction of silver in the roasted ore was, by the use of the Russell solution, brought up to 86.74 per cent., or 7.81 per cent. more than the amount shown to have been present in the roasted ore as chloride, proved conclusively that, whatever form the chloride had been converted into during the process of washing, it was soluble in the cuprous hyposulphite solutions of the Russell process. This led the writer to the conclusion that the silver chloride had been converted to a sulphide, which is readily soluble in the Russell solution."

The force of this reasoning is not apparent. Assuming that this extra 7.81 per cent. of silver was in the state of sulphide when dissolved (though there seems to be no proof of that except the fact that the sulphide is soluble in Russell solution), what is there to show that it had previously been a chloride? Mr. Morse's statement (confirmed by his tables) is that this silver was *not* present as chloride in the roasted ore before washing. How, then, can he suppose it to have been a chloride, and to have been altered to a sulphide by the reaction he describes? Why may it not have been a sulphide, or some one of the many other compounds of silver soluble in the Russell solution? In short, Mr. Morse's theory is clearly not applicable to this silver, which was not shown as chloride in the roasted ore. In other words, the excess of 7.81 per cent., to which he refers, is not a result of any "going-back" of chlorination, and therefore cannot offer any proof as to the cause of such "going-back." On the other hand, the figures which are pertinent to that discussion are those of the final column of his Table I., which it is therefore of interest to examine.

It will be noticed at a glance, that a marked change occurs in the last column of this table, with the treatment of mixture

No. 9.* In fact, the change should be evident in the line above, where, by a typographical error, the decrease of silver chloride by roasting is stated as 13.70 per cent., whereas, the subtraction performed upon the figures in the two preceding columns shows it to have been only 8.6 per cent.† From this point down the column, ending with No. 19 (a period of nearly twelve months), the improvement and the uniformity of the results are (with two prominent and two lesser exceptions) quite out of proportion to the record preceding No. 8, and vastly greater than can be accounted for by the difference in the zinc-sulphide of the ore. The explanation is simple, and will be given later. The fact is merely noted here, as throwing doubt at the beginning upon Mr. Morse's theory of the cause of "going-back."

A similar doubt is raised by a study of separate groups of the mixtures of Table I.‡

I.—*Groups Showing Practically the Same Amount of Zinc as Sulphide in the Roasted Ore.*

No.	Raw Ore in Mixture. Tons.	Total Ore in Group. Tons.	Zinc Sulphide in Roasted Ore. Per cent.	Decrease of Silver Chloride by Washing. Per cent.
13	2380		1.14	9.18
10	2656	6101	1.15	0.55
18	1065		1.25	12.59
1	1309		2.00	22.30
5	1276	5020	2.04	16.14
6	2435		2.07	19.58
8	2123		0.82	8.6
17	410	4566	0.89	4.0
14	2033		0.97	0.24

Total amount of raw ore represented in the above set of groups, 15,687 tons.

* Mixture No 9, representing 2687 tons of ore, was the work of August. I left Aspen the first day of September.

† NOTE BY THE SECRETARY.—This error in Table I., and the corresponding error in Table II., have been corrected by Mr. Morse for the permanent publication of his paper in the present volume.

‡ For this purpose, I use only 12 of the 19 mixtures; my object in the re-arrangement being, not to deduce a general law, but to inquire whether the discrepancies which Mr. Morse considers "slight," and hence not contradictory of his theory, are really insignificant.

II.—*Groups Showing Practically no Variation in Decrease of Silver Chloride by Washing.*

No.	Raw Ore in Mixture. Tons.	Total Ore in Group. Tons.	Zinc Sulphide in Roasted Ore. Per cent.	Decrease of Silver Chloride by Washing. Per cent.
12	2602	4982	1.64	9.66
13	2380		1.14	9.18
11	2860	5547	0.68	3.18
9	2687		1.00	3.32

Total amount of raw ore represented in the above set of groups, 10,529 tons; total in both sets, 26,216 tons.

In the first set of groups we would naturally expect to find, on Mr. Morse's theory, a uniform decrease of silver chloride by washing. The reverse is the case.

In the second set of groups, the decrease of silver chloride by washing being very uniform, we ought to find equal uniformity in the amount of zinc present as sulphide in the roasted ore; but the reverse is again the case.

The groups showing these discrepancies comprise 26,216 tons, or, deducting No. 13, which has been used twice, 23,836 tons out of the 35,144 tons covered by Table I. I think it clear, therefore, that Mr. Morse's data do not support his theory; that is, they indicate some other cause, besides the percentage of zinc-blende in the ore, for the "going-back" of chlorination. The particulars of Mr. Hoyt's laboratory-experiments not being given, it is impossible to examine them critically.

With regard to the experiments of Mr. Frank A. Bird, the accuracy of which is unquestioned, it should be observed that the conditions were totally different from those at Aspen. In his experiments from 55 to 58 per cent. of zinc was present; in the Aspen ore, from 1 to 3 per cent.

The difference in the strength of the brine solution between the two cases is equally great. Moreover, for the months of May, June and July at Aspen, represented by mixtures Nos. 7 and 8, the percentage of salt in the ore (12.7 and 12.0 per cent.) was the highest for the year, while the amount of silver chloride dissolved and extracted in July was 20.9 per cent., being exceeded by only one month during the year. The results of these

lots as to decrease in silver chloride by washing do not conform with those deduced from Mr. Bird's experiment.

Table IV. does not at all establish Mr. Morse's contention, unless he first proves that the extra silver, insoluble in "hypo"-but soluble in the Russell solution, was present in the washed ore as a sulphide. Any one of the many other compounds of silver soluble in the Russell solution would answer equally well.

The direct cause of the remarkable change shown in Table I. and beginning, as I have explained, with mixture No. 8, was the admission into the Stetefeldt furnace of all the air possible, thus oxidizing the sulphur more completely in the furnace, and leaving less work to be done on the cooling-floor, where the formation of much sulphurous acid is otherwise unavoidable.

As a further proof that the cause of the above change was known to me while at Aspen, I quote from "A Review of the Russell Process" as follows:

"It was also noticed at Aspen that some of the lowest losses in silver were experienced during the months when a heavy excess of sulphur had been used and only a limited supply of air allowed to enter the furnace. It was afterwards observed that additional air produced a higher chloridization of the silver, but that the losses by volatilization were also higher."

On the action of oxygen in chloridizing-roasting, I may have something further to say in a subsequent paper.

Speaking of the precipitation of the silver in the wash-water by means of precipitated base metal sulphides, Mr. Morse says, "The result of turning this reaction to practical account will be of interest." In this connection, I quote the following from my "Review of the Russell Process:"

"In the precipitation of the wash-water a curious fact was noticed, which, while not new to chemistry, had probably never before been made use of in practice. The sulphides obtained from the wash-water containing the soluble salts were of a very low grade, frequently not exceeding 2000 ounces of silver per ton. The writer, several years ago, made a few laboratory-experiments with the precipitated sulphides of copper, lead and iron on chloride of gold solution, and found that all these precipitated the gold in the form of sulphide. Percy states that certain of the base metal sulphides act in a similar manner towards chloride of silver. With this idea in view, the tanks containing the wash-water sulphides were allowed to stand for six or seven weeks before taking out any of the precipitates. All of the wash-water during this time was precipitated in these tanks. After each precipitation the sulphides were allowed to settle and the solution carefully decanted off. When the sulphides were finally taken out and dried, sampled and assayed, they were found to contain nearly 7000 ounces of silver to the

ton, showing conclusively that a reaction had been going on by which part of the metals of base sulphides were gradually converted into soluble chlorides for which an equivalent amount of silver was precipitated."

The above precipitation, however carefully done, did not save all the silver, and the precipitation therefore was not complete. Considerable silver was saved from the decanted solution in a subsequent precipitation with scrap-iron and other precipitants.

The mere questions of personal priority involved in the foregoing criticisms of Mr. Morse's paper have, of course, little importance. The real question, which we are both equally interested to answer, is, whether the theory he supports is sufficient to account for the facts, and whether the facts both sustain it and exclude any other. As to this question, I have frankly stated my dissent, and I do not feel that Mr. Morse has effectually either established his own view or overthrown mine.

Prof. H. O. Hofman, Boston, Mass. (communication to the Secretary): Mr. Morse's paper and Mr. Godshall's discussion of it recall to my mind three articles by Mr. Dubois, Mr. Gmehling and the late Mr. Aaron on the effect of sulphides on silver held in solution by sodium hyposulphite. Mr. A. H. Dubois,* in making a solubility-test with sodium hyposulphite on raw ore, assaying 320 ounces of silver per ton, found that if he filtered fifteen minutes after starting, he extracted from 65 to 70 per cent. of the silver, while if he left ore and solvent in contact for sixteen hours before filtering, he obtained only 12 per cent. Working synthetically, he dissolved 800 milligrammes of silver in the form of silver chloride in 250 c.c. of sodium hyposulphite, added 4 grammes of sulphurets (galena, blende and pyrite) containing 3 milligrammes of silver, and left them in contact for sixteen hours, shaking repeatedly. The sulphurets, after filtering and washing with sodium hyposulphite and water, were assayed, and gave 282 milligrammes of silver.

Mr. C. H. Aaron† found that the solubility assay of a roasted silver-ore gave much lower results if solvent and ore had remained in contact over night than if only a short time. He found by experiment that copper and silver are quickly pre-

* *Min. and Sci. Press.*, May 11, 1889, p. 334.

† *Min. and Sci. Press*, May 25, 1889, p. 374; *Eng. and Min. Jour.*, June 22, 1889, p. 563.

cipitated by zinc sulphide from hyposulphite solutions; also that from a dilute solution of potassium cyanide silver was precipitated by zinc sulphide, and that in a concentrated solution the silver sulphide soon redissolved.

In experimenting at Huanchaca, Bolivia, with a siliceous silver-ore, which contained much blende in addition to other metallic sulphides, Mr. Gmehling* found that if he leached the chloridized ore as it came from the furnace, *i.e.*, as coarse 10-mesh stuff, he obtained a higher extraction than if he first passed it through a 40-mesh sieve. The explanation for this is, that in quick filtering, undecomposed metallic sulphide (probably blende), or sulphate (probably lead sulphate), in the ore remains a shorter time in contact with the silver-solution, and has therefore a less harmful effect.

The Monazite Districts of North and South Carolina.

Discussion of the Paper of Mr. C. A. Mezger (see p. 822).

(Atlanta Meeting, October, 1895.)

R. W. Raymond, New York City: It seems questionable to me whether Mr. Mezger's identification of the rock-structure he describes, as the *Augengneiss* of previous authors, is warranted by the definitions given by them. Naumann (1854)† describes *Augengneiss* as a porphyry-like variety, "which has assumed, by reason of single large kernels or granular concre-

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* *Oesterr. Zeitsch. für Berg und Hüttenwesen*, 1890, p. 284.

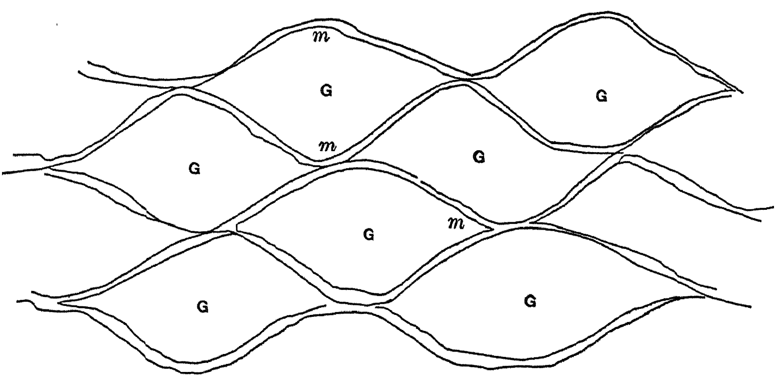
† *Lehrbuch der Geognosie*, Leipzig, 1854, vol. ii., p. 77. On p. 120 of the same volume, the author, speaking of the transitions of mica-schist into gneiss, says:

"The transition to gneiss is not infrequently to be regarded as the result of the metamorphic action of adjacent masses of granite. . . . . The mica schist becomes first finely and then more coarsely *flaserig* gneiss. This passes gradually, by the incorporation of individual lenses of feldspar, into a distinctly coarse *flaserig* structure; the feldspar-lenses gradually thicken to the size of hazel-nuts or walnuts, becoming at the same time more numerous, and thus is formed the so-called *Augengneiss.*"

The structure called *flaserig* the same author defines (vol. i., p. 480) as characterized by "thin, short layers, or lenticular masses, of granular nature, alternating with still thinner, short and somewhat curved layers (*Flasern*) of scaly structure, which cling to the former in a parallel arrangement."

tions of feldspar, a porphyritic structure, and, when these feldspar masses have a thick lenticular form, is called also *Augengneiss*." This author names as localities Sweden, Norway, the Hebrides and Connecticut, but not Brazil. Geikie (1893)* speaks of "porphyritic gneiss or *Augengneiss*, in which large eye-like kernels of orthoclase or quartz are dispersed through a finer matrix, and represent larger crystals or crystalline aggregates, which have been broken down and dragged along by shearing-movements in the rock." But I do not find in his work, or in any other, a distinct citation of the rocks of Rio de Janeiro as typical *Augengneiss*. It will be observed, however, that both Naumann and Geikie (taken by me as representing both early and recent authorities on the subject) specifically de-

Fig. 1.



Structure of Gneiss at Rio de Janeiro, and at Claremont, N. C.; G, granitic lenses; *m*, gneiss, mica or mica-schist.

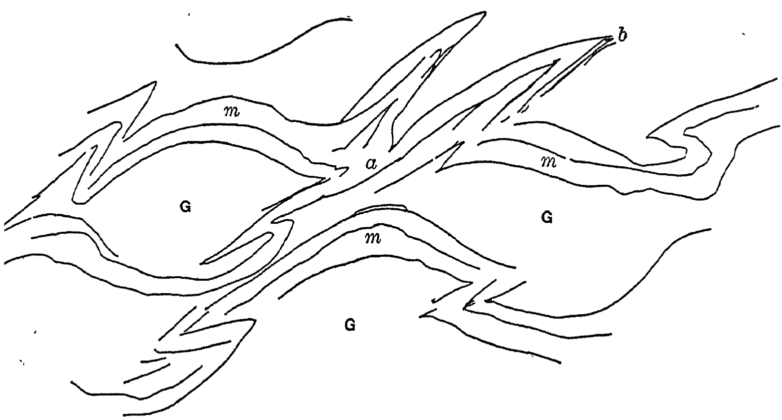
scribe the granular inclusions in *Augengneiss* as consisting of single minerals (quartz or feldspar) only. This raises the question, whether the structure observed by Mr. Mezger can be classed at all under that name. According to his description, it exhibits inclosures of granite, not merely of single minerals which occur in granite.

Prof. Geikie's theory as to *Augengneiss* evidently is, that the rock was originally formed by cooling from igneous fusion; that the gneissic structure was subsequently imposed upon it; and that the kernels or lenticular inclosures represent original

* *Text-book of Geology*, London and New York, 1893, p. 186.

masses, resulting from cooling and crystallization, which have resisted the subsequent schistification. If this theory be correct, it is conceivably applicable as well to residual lenses of granite as to residual masses of crystalline quartz or feldspar simply; and this consideration raises the further question, whether the structure described by Mr. Mezger, though not strictly falling under the previous definition of *Augengneiss*, may not be a result, on a larger scale, of the same operating causes. An affirmative answer to this question would have this important corollary, that geologists could regard the lenticular structure referred to as indicating an originally igneous rock.

Fig. 2.



Structure of Gneiss at Morgantown and Shelby, N. C.; G, granitic enclosures; *m*, gneiss, mica or mica-schist.

Mr. Mezger: As to the name to be given to the rocks in question, I took the term *Augengneiss* from my friend, the late Prof. Stelzner, of Freiberg, with whom I discussed the subject several times after my return from Brazil. But the definitions given by Naumann (who evidently did not refer to the Brazilian rock) and by Geikie, as quoted by Dr. Raymond, do not at all fit the rocks of Rio or of North Carolina. In the *Augengneiss* which they describe, the proportion of gneiss to inclosures is more than 2 : 1. In the rock to which I refer, the granite is by far the greater mass, often 10, and sometimes 100, times as much as the mica, gneiss or mica schist surrounding it. Fig. 1 shows a section of the Rio rock, which may serve also as a section of the rock at Claremont, N. C., except as to dimen-

sions. At Rio, the thickness of the granitic lenses is 4 to 6 inches, and the length 6 to 10 inches, the grains of quartz and feldspar being about $\frac{1}{4}$-inch in diameter, and the layers of gneiss or mica are only from 1 to 2 millimeters (0.04 to 0.08 inch) thick. The whole mass is so homogeneously solid that the rock can be used for door-steps, fence- and door-posts, etc. It is, in fact, the only material at hand in Rio de Janeiro which is cut for building or ornamental purposes, and it meets the eye everywhere. The mica or gneiss layers often appear like faint lines, but are never absent.

At Claremont these layers are about an inch thick and fairly regular. The lenses are about 1 foot thick and 18 inches long. The regularity of these lenses is maintained only so long as their thickness does not exceed about 2 feet. When they become larger, they assume, as their first irregularity, zigzag lines in the gneiss-sheets, as indicated in Fig. 2. This structure may be seen at Morgantown, and by the railway station at Shelby, N. C. The length, *a–b*, is often 6 to 10 feet, and even more, increasing with the size of the lenses. Of course, the whole mass thus becomes an apparently irregular and indefinite mixture of gneiss and granite; and any one seeing these rocks first, without having previously studied the smaller and more regular lenses, would naturally be unable to discern any law of structure. Close investigation has always (but not always easily) shown me the existence of the structure above described.

In view of these circumstances, and of the further fact that the inclosures in *Augengneiss*, as defined by the authors quoted by Dr. Raymond, are of single minerals, and not of granite, it would probably be better to give a distinctive name to the structure I have pointed out. Perhaps *Lens-gneiss* would describe it best.

I have no explanation to offer at present of the structure presented by the Rio gneiss, which is the simplest and most regular type. But I cannot accept the view that the granite lenses are unaltered remnants of an original granitic mass, which Geikie asserts for the porphyritic *Augengneiss*. Even for that rock, it does not seem to me an acceptable hypothesis; for the lens-gneiss it is clearly inadequate.

DR. RAYMOND: I must agree with Mr. Mezger that the structure described (and now more fully illustrated) by him is

not to be, without further investigation, directly connected with that of *Augengneiss*, for which the term "porphyritic gneiss" seems to have been accepted as a synonym. Nobody would call the structure shown in his illustration "porphyritic." As to the term "lens-gneiss," which he suggests, I can say only, that nothing better occurs to me, and yet that it seems somewhat illogical and misleading to class as a variety of gneiss a rock which is, according to his description, nearly all granite, without gneissic structure. The matter of nomenclature, however, may be fairly considered a subordinate one, and left for final settlement until the difficult pending question of the relations between granite and gneiss, so-called, shall have been more definitely answered. In the solution of that problem, the acute and careful observations of Mr. Mezger will certainly have to be taken into account.

INDEX.

[NOTE.—In this Index the names of authors are printed in small capitals, and the titles of papers in italics. Casual references, giving but little information, are usually indicated by bracketed page-numbers.]

ERRATA.

On pages 118, 120, 126 and 127, "Ramsey" should be "Ramsay."
On page 670, line 9 from bottom, "Randolph county" should be "Davidson county."
On page 682, line 5 from top, after "Montgomery" read "(now in Stanley)."
On page 686, line 8 from top, "Randolph county" should be "Montgomery county."
On page 718, line 8 from bottom, "Thompson" should be "Thomson."
On page 751, line 13 from bottom, "Blackmore" should be "Blackmer."